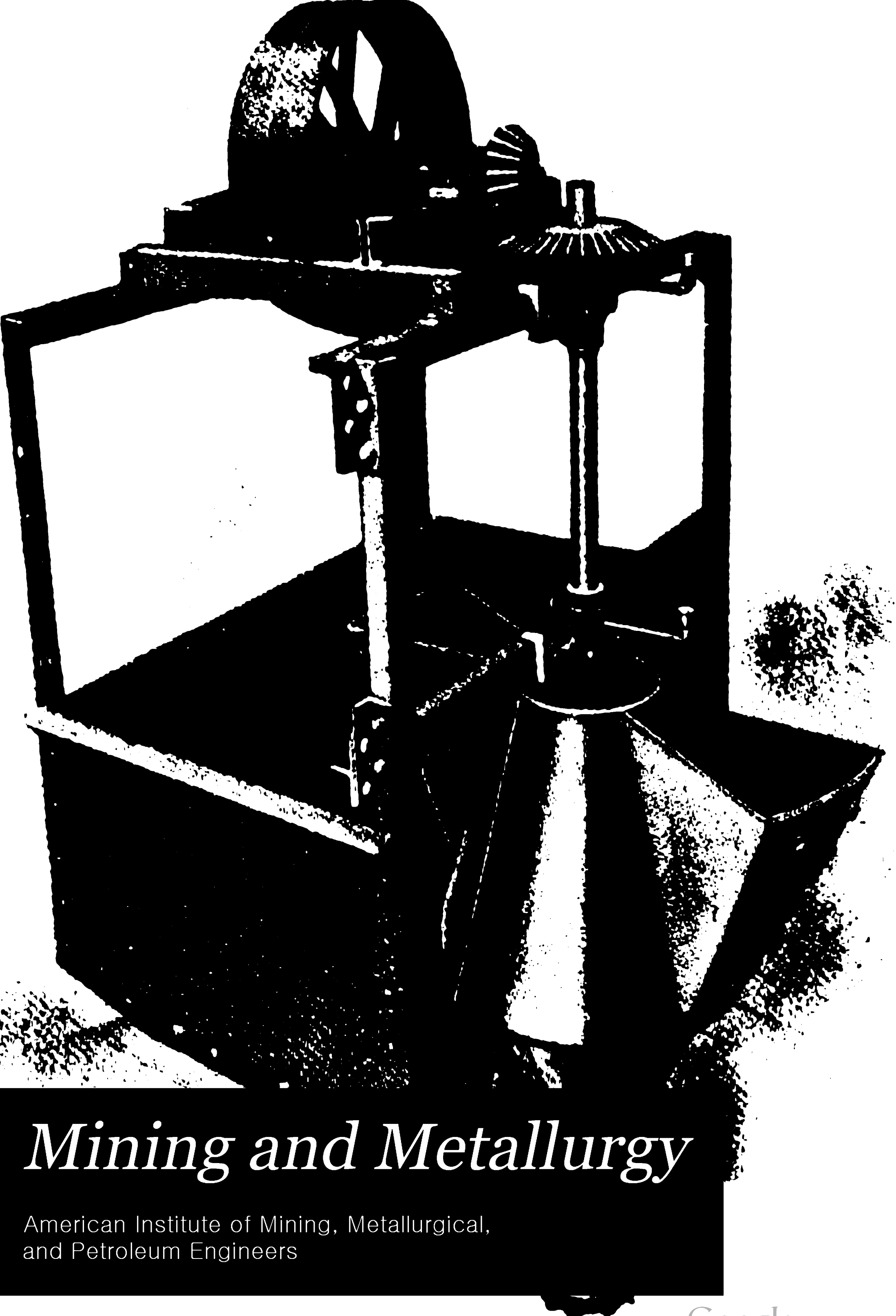

Mining and Metallurgy

American Institute of Mining, Metallurgical, and Petroleum Engineers

No. 31 JULY 1909

Bulletin of the American Institute of Mining Engineers.

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EDITORIAL OFFICE AT 29 WEST 39TH STREET, NEW YORK, N. Y.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.

No. 31 JULY 1909

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SECTION I.—INSTITUTE ANNOUNCEMENTS.

This section contains announcements of general interest to the members of the Institute, but not always of sufficient permanent value to warrant republication in the volumes of the *Transactions*.

SECTION II.—TECHNICAL PAPERS AND DISCUSSIONS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

A detailed list of the papers contained in this section is given in the Table of Contents. They have been so printed and arranged (blank pages being left when necessary) that they can be separately removed for classified filing, or other independent use.

A small stock of separate pamphlets, duplicating the technical papers given in Section II. of this Bulletin, is reserved for those who desire extra copies of any single paper.

Comments or criticisms upon all papers given in this section, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

All communications concerning the contents of this Bulletin should be addressed to Dr. Joseph Struthers, Assistant Secretary and Editor, 29 W. 39th St., New York, N. Y. (Telephone number 4600 Bryant).

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For the year ending February, 1910.

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

INSTITUTE ANNOUNCEMENTS.

The Bulletin.

As already announced in the January *Bulletin*, this publication will be issued during the coming year monthly instead of bi-monthly as heretofore. Among other reasons for this change, it is desired to effect thereby the earlier transmission to members of important papers, lists of candidates for membership, and other items of timely information.

Spokane Meeting and Excursions.

Further details of the 97th meeting of the Institute have been sent to members in the Circular of May 8, 1909.

For convenience of the members a summary of the additional information is here presented.

A special train of Pullman cars will leave Chicago about September 15. This train will be side-tracked at all places visited, and the sleepers will be available for lodging and the dining-car for meals during the entire trip, except during the Yellowstone Park tour.

The general cost of the trip per member will be $300, which includes transportation, berth, and meals for the entire trip from Chicago back to Chicago, almost 6,000 miles, and the tour through Yellowstone Park, occupying in all about 30 days. Special accommodations on the train will be furnished at the following rates:

Chicago to Chicago, including transportation, berth, and meals,	$300.00
State-room for one person, $200.00 extra, or	500.00
State-room for two persons, total,	700.00
Drawing-room for two persons, total,	800.00
Drawing-room for three, total,	1,100.00

The foregoing schedule depends upon a sufficient number of passengers taking part in the entire excursion to permit arrangements for a special train, and members and guests who anticipate taking part in the excursion are therefore earnestly urged to notify Mr. Dwight as soon as possible.

The excursions connected with the Spokane Meeting will have much of professional interest: the varied formations in the Yellowstone Park; the great copper-plants at Butte and Anaconda; the mining and metallurgical enterprises at Cœur d'Alene; the mining exhibits at the Alaska-Yukon-Pacific Exposition, Seattle; the Tacoma Smelter at Tacoma; and the mining and metallurgical operations in the vicinity of Salt Lake, at Bingham, Murray, and Garfield; and (possibly) the steel-plant and lead-smelter at Pueblo.

The scenic interest can be appreciated by mentioning the Yellowstone Park; the Cœur d'Alene district; the daylight ride from Spokane to Seattle; the Cliff drive at Tacoma; the bold scenery of the Wasatch mountains, and the Royal Gorge. It would be very difficult to plan a scenic trip of greater variety and beauty.

Members intending to visit the meeting of the American Mining Congress at Goldfield, beginning about October 15, may be interested to know that a report on the geology and ore-deposits of the Goldfield District will be published probably in July by the U. S. Geological Survey as *Professional Paper No.* 66. This report may be obtained by applying to the Director of the U. S. Geological Survey, Washington, D. C.

The following list comprises the names of members and guests who have reserved accommodations for the entire trip:

Mr. and Mrs. V. M. Braschi.
Mr. and Mrs. Owen Brooke.
Miss Brooke.
Mr. and Mrs. D. R. C. Brown.
Mr. and Mrs. D. W. Brunton and guests.
Mr. and Mrs. F. J. Campbell.
Mr. Charles Catlett.
Mr. and Mrs. H. S. Chamberlain.
Mr. and Mrs. W. B. Cullum.
Mr. Theodore Dwight.
Mr. Anton Eilers.
The Misses Eilers.
Miss Anna M. Fries.
Mr. T. B. Greenfield.
Mr. and Mrs. M. H. Harrington.
Mr. A. Harrington.
Miss Harrington.
Mr. and Mrs. L. Holbrook.
Mr. and Mrs. E. S. Hutchinson.
Mr. and Mrs. William Kelly.
Prof. and Mrs. William Kent.
Mr. Charles Kirchhoff.
Mr. and Mrs. John Lilly.
Mr. and Mrs. James E. Little.
Mr. E. H. Messiter.
Mr. and Mrs. J. W. Nesmith.

Mr. and Mrs. E. W. Parker and guest.
Mrs. E. Pechin and guest.
Mr. and Mrs. W. S. Pilling.
Miss Pilling.
Master Pilling.
Mr. F. D. Rash.
Dr. and Mrs. R. W. Raymond.
Miss Righter and guest.
Mr. W. L. Saunders.
Mr. W. L. Saunders, Jr.
Miss Saunders.
Miss E. Saunders.
Miss J. Saunders.
Mr. A. T. Shurick.
Dr. Joseph Struthers.
Mr. and Mrs. Tarwater and guests.
Mr. and Mrs. A. E. Vaughan.
Mr. S. T. Wellman.

The list of those intending to go on the excursion is filling up, and since preference in position of state-room or berth is given in the order of application, members and guests who expect to accompany the party and have not yet communicated with Mr. Theodore Dwight, 29 West 39th St., New York, N. Y., are earnestly requested to do so without delay. ALL applications should be in by July 30, 1909.

Meetings of Other Societies.

International Congress for Mining, Metallurgy, Applied Mechanics, and Practical Geology, Düsseldorf, 1910.—At the closing session, July 1, 1905, of this Congress, held in connection with the Lüttich Exposition of that year, it was resolved to accept the invitation extended by the mining industries of Rheinland-Westphalia, and to hold the next Congress in that region.

In accordance with this resolution, the Congress will be called to meet in the latter part of June, 1910, at Düsseldorf. For this meeting, which will occupy about one week, extensive preparations are in progress, including visits to technical institutions and industrial establishments, excursions to geologically-interesting localities, etc., intended to illustrate the papers and addresses presented in the four sections above named.

The exact date, together with more detailed and definite particulars of the meeting, will be published later. Meanwhile, inquiries, suggestions, and announcements of proposed papers may be addressed to Dr. E. Schrödter, Chairman of the Committee of the Congress, Düsseldorf, Jacobistrasse 5, Prussia.

Office Facilities for Visiting Members.

A separate room in the suite occupied by the American Institute of Mining Engineers on the ninth floor of the United Engineering Society Building, has been equipped with furniture and telephone extension for the temporary use of members of the Institute or of sister societies, or visitors suitably accredited.

Members of the Institute visiting New York for a short time, who need office facilities during their stay, or members residing in the city who need temporary office accommodation, can arrange to have set apart for their exclusive use a room, equipped with office furniture, telephone, etc., in the suite of the Institute. It is not the intention to give possession of the room to any individual for an indefinite time, but to offer to members of the Institute an opportunity to acquire a well-located, well-equipped business headquarters to carry on transactions which would not warrant the establishment of a permanent office. The room devoted to this purpose is entirely separate from the reception- and writing-rooms for the general use of the members. A small fee will be required for the use of the facilities furnished. For the conditions of this privilege, inquiry should be made at the office of the Secretary of the Institute.

How to Use the "Transactions" of the Institute.

Buy a copy of the Complete Analytical and Alphabetical Index of Volumes I. to XXXV., inclusive.

If you own a full set of the *Transactions*, this Index will make the whole of it instantaneously available without detailed research into each volume separately.

If you do not own such a set, this Index will be even more valuable, for it will show you what particular papers you need to know more about, and perhaps to study. Thus, any person possessing this Index can ascertain at once what has been published in the *Transactions* on a given question, and can learn, by writing to the Secretary, what is its nature, whether it is still to be had in pamphlet form, where it can be consulted in a public library, at what cost it can be copied by hand, etc., etc.

In short, to those who own complete sets of the *Transactions*, this Index will be a great convenience; but to those who do not, it will be a professional necessity.

This volume is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. It has not been stereotyped, and the edition is limited to 1,600 copies. The price of the volume, bound in cloth, is $5, and bound in half-morocco to match the *Transactions*, $6. The delivery charges will be paid by the Institute on receipt of the above price.

Hydrographic Chart.

Owing to the great value to hydrographers of the chart contained in the paper, A Graphic Solution of Kutter's Formula, by L. I. Hewes and Joseph W. Roe (*Bulletin No.* 29, May, 1909, p. 454), a special edition for office or field use has been printed on durable cloth. Copies of this separate chart may be obtained, at a cost of 50 cents each, on application to the office of the Secretary of the Institute.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

RULES.

For the protection and convenience of members, the following rules have been adopted:

The Secretary of each Society will, upon application, issue to any member of his Society in good standing a personal, non-transferable card, entitling him to the use of the Libraries in the alcoves of the Reading-Room.

This card, as well as any card of introduction given to a non-member, must be signed by the person receiving it, and surrendered at the desk at the time of its presentation. At every visit he must identify himself by signing his name in the registry.

Strangers who desire to enjoy the privilege of entering the alcoves are requested to present either letters of introduction from members, or cards, such as will be furnished upon application by the Secretary of each Society. The first two alcoves are free to all; and admission to the inside alcoves is given upon proper introduction.

The above rules apply to all persons except officers of the three Societies, personally known as such to the librarians.

The librarians are not permitted to lend to any person any catalogued pamphlet or volume, unless authorized in writing so to do by the Secretary or Chairman of the Library Committee of the Society to which the pamphlet or volume belongs.

Any person discovering a mutilation or defect in any book of the libraries is requested to report it to the librarian on duty.

Library Additions.

From May 15 to June 15, 1909.

ARCHITECTS AND BUILDERS POCKET-BOOK. Ed. 15. By F. E. Kidder. New York, J. Wiley & Sons, 1908. (Gift of Publishers.) Price, $5.

[SECRETARY'S NOTE.—This is the fifteenth edition of the well-known work first issued in 1884 by the late Frank E. Kidder. In 1904, twenty years later, Mr. Kidder prepared the fourteenth edition, by a thorough revision with many additions; and since his death this new edition has been called for. It was found necessary to make numerous changes in order to keep pace with modern improvements in construction, etc., and for this purpose Chapters XXIII. and XXIV., Part II., dealing with the Fire-Proofing of Buildings and Reinforced Concrete, have been re-written by Rudolph P. Miller, long connected with the Department of Buildings in New York City. These subjects, together with that of Paints and Varnishes (which has been brought up to date by Prof. Alva H. Sabin), are now presented so as to exhibit the latest and best practice.

During the quarter-century of its existence Kidder's Pocket-book has outgrown its title, the original volume of 586 pages having become one of 1,661 pages, too large to be conveniently carried in the pocket; but, as the author observed in the last preface he ever wrote, "As experience has shown that the book is used principally at the desk or draughting-table, and is seldom carried in the pocket, it is believed that the convenience of having everything in one book will more than offset any disadvantage resulting from increase in bulk." It is unnecessary to add a recognition of the merits of the work. One does not recommend the classics.—R. W. R.]

BEST TYPE OF CANAL FOR PANAMA. (From *Engineering News*, Feb. 25, 1909.) (House Document No. 10, Pt. 2.) Washington, U. S. Government, 1909. (Gift.)

BIBLIOGRAPHIE DER DEUTSCHEN ZEITSCHRIFTEN LITERATUR. Vol. 23. By F. Dietrich. Leipzig, 1909. (Purchase.)

BOSTON PUBLIC LIBRARY. Annual Report of the Trustees, 57th. Boston, 1909. (Exchange.)

BRAUNKOHLE. Year 8, No. 7–date. Halle a. S., 1909–date. (New exchange.)

BROWN'S DIRECTORY OF AMERICAN GAS COMPANIES, 1909. New York, 1909. (Gift of the *Progressive Age*.)

[SECRETARY'S NOTE.—This comprehensive and indispensable directory, established in 1887, has assumed a character scarcely contemplated at that time. The competition between illuminating-gas and electric lighting, instead of displacing the former, has led to new materials, economies, and improvements in its manufacture, and new fields of application for gas (natural or artificial) as fuel—such as the production of the Welsbach light and the use of gas-engines as motors, even for the generation of electric currents, and for a thousand wider uses. Moreover, a host of strong gas-companies, old and new, have "electric departments," so that they can furnish either form of light or power, as circumstances require. The resulting benefit to the public—due not to legislation but to commercial competition and enterprise—is indicated by the astonishing exhibit of the gas-industry made in the pages of this Directory.

The book contains a detailed list of gas companies in all the States of the Union, the British-American provinces, Mexico, and South America, giving particulars as

to the capitalization, bonded debt, officers, process of manufacture, cost of raw material, price of product, extent and nature of business, technical details of operation, etc., etc. These items are more fully given in some cases than in others, bnt the completeness of the data furnished by the Directory as a whole is surprising. There are also lists of acetylene and of gasoline town-plants, the number of which is likewise surprising.

The Appendix contains still more detailed descriptions of a score of the more important plants and operations in the United States, and there is a list of the gas-associations of the country, comprising not only commercial bodies, but also many active and useful societies of gas-engineers, intent upon the technical progress of their art, and worthily representing what has become practically a new profession.—R. W. R.]

BRITISH ASSOCIATION FOR THE ADVANCEMENT OF SCIENCE. Winnipeg Meeting, Aug. 25 to Sept. 1, 1909. Visit to Cobalt and Sudbury. Toronto, L. K. Cameron, 1909. (Gift.)

CANADA—MINES DEPARTMENT. Annual Report on the Mineral Production of Canada, 1906. Ottawa, S. E. Dawson, 1909. (Exchange.)

CANADA—MINES DEPARTMENT. Preliminary Report on Gowganda Mining Division, District of Nipissing, Ontario. By W. H. Collins. Ottawa, C. H. Parmelee, 1909. (Exchange.)

CANADA—MINES DEPARTMENT. Preliminary Report on the Mineral Production of Canada in 1908. Ottawa, Government, 1909. (Exchange.)

CANADA—MINES DEPARTMENT. Summary Report of the Geological Survey Branch of the Department of Mines of Canada, 1908. Ottawa, C. H. Parmelee, 1909. (Exchange.)

CANADIAN MINING INSTITUTE. List of Officers, Members, and Student Members, Mar. 31, 1909. Montreal, 1909. (Exchange.)

CATALOGUE OF THE TYPE-SPECIMENS OF MAMMALS IN THE UNITED STATES NATIONAL MUSEUM, INCLUDING THE BIOLOGICAL SURVEY COLLECTION. (Bulletin No. 62, U. S. National Museum.) By M. W. Lyon, Jr., and W. H. Osgood. Washington, U. S. Government, 1909. (Exchange.)

COAL AND COKE. Vol. 16, No. 10–date. Baltimore, 1909–date. (New exchange.)

COMITÉ GÉOLOGIQUE DU RUSSIE. Bulletin. Vol. 26, Nos. 1–4, 8–10; Vol. 27, Nos. 2–3. St. Petersburg, 1907–1908. (Exchange.)

——— MÉMOIRES. Nouvelle série, Livraison 28, 30, 37, 38, 41, 42. St. Petersburg, 1908. (Exchange.)

COMPLETE MINERAL CATALOGUE. Ed. 12, revised. By W. M. Foote, Philadelphia, 1909. (Gift of Dr. R. W. Raymond.)

COMPLETE YEAR BOOK OF WISCONSIN, ILLINOIS, AND IOWA LEAD AND ZINC MINES, 1909. Dubuque, 1909. (Gift of *Iowa Farmer* Publishers.)

CONTRIBUTIONS FROM THE UNITED STATES NATIONAL HERBARIUM. Vol. 12, Pts. 5–9. Washington, U. S. Government, 1909. (Exchange.)

COPPER HANDBOOK. Vol. VIII. Compiled by H. J. Stevens. Houghton, 1908. (Purchase.)

CRITICAL SUMMARY OF TROOST'S UNPUBLISHED MANUSCRIPT ON THE CRINOIDS OF TENNESSEE. By Elvira Wood. (Bulletin No. 64, Smithsonian Institution, U. S. National Museum.) Washington, U. S. Government, 1909. (Exchange.)

CUBA—POPULATION, HISTORY, AND RESOURCES, 1907. Washington, U. S. Government, 1909. (Gift of U. S. Bureau of the Census.)

DEUTSCHEN NORMALPROFIL-BUCH FÜR WALZEISEN ZU BAU-UND SCHIFFBAU-ZWECKEN. Erster Band. Aachen, 1908. (Gift.)

——— SIEBENTE VERBESSERTE AUFLAGE. Aachen, 1908. (Gift.)

EXPLORATIONS IN ALASKA, 1899, FOR AN ALL-AMERICAN OVERLAND ROUTE FROM COOK INLET, PACIFIC OCEAN, TO THE YUKON. By J. S. Herron. Washington, U. S. Government, 1909. (Gift of U. S. War Department.)

GATUN DAM AND EARTH DAMS IN GENERAL. (From *Engineering News*, Apr. 1, 1909.) (House Document No. 10.) Washington, U. S. Government, 1909. (Gift.)

GEOLOGICAL MAP OF PORTIONS OF HASTINGS, HALIBURTON, AND PETERBOROUGH COUNTIES, PROVINCE OF ONTARIO. 1905. (Exchange.)

GEOLOGY AND WATER RESOURCES OF THE HARNEY BASIN REGION, OREGON. (Water-Supply Paper No. 231, U. S. Geological Survey.) By G. A. Waring. Washington, U. S. Government, 1909. (Exchange.)

GOLDEN MILE DEVELOPMENTS, 1908. Descriptive of the Underground Workings of the Kalgoorlie Gold-Mines. Perth, 1909. (Exchange.)

HUDSON BAY FROM YORK FACTORY TO FORT SEVERN, MAP OF PART OF SOUTH-WESTERN COAST, 1905. (Exchange.)

ILLINOIS WATERWAY REPORT, WITH PLANS AND ESTIMATES OF COST FOR A WATERWAY FROM LOCKPORT, ILLINOIS, TO UTICA, ILLINOIS, BY WAY OF THE DESPLAINES AND ILLINOIS RIVERS, 1909. Springfield, Illinois State Journal Co., 1909. (Gift of Internal Improvement Commission of Illinois.)

INTERNATIONAL YEAR BOOK, 1908. New York, Dodd, Mead & Co. 1909. (Gift of Publishers.)

[SECRETARY'S NOTE.—This year-book is the second of the series issued by the publishers as annual supplements to their excellent International Cyclopedia. Such supplementary volumes are not only useful to general students, but necessary to those who wish, as we all do at times, to revive and vivify recent personal recollections. Who has not felt the annoying difficulty of fixing a vaguely remembered fact not yet old enough to be reliably recorded in the latest history or cyclopedia? What rummaging in *Poole's Index*, or in the files of newspapers, have we not performed (too often in vain) for such a purpose.

The year 1908 was peculiarly rich in events significant of national and international change and of scientific and technical progress—in notable things achieved and even greater things projected and begun. In other words, it presented an immense number of live questions and movements, concerning which we are still talking and thinking, and desiring to refresh our memories. To meet this situation, material which could be easily found elsewhere has been omitted from the volume, and fresh contributions from official sources or recognized authorities have been embodied in a work which fairly deserves the claim of its editors that it is a cyclopedia for the year, not only "up-to-date," but free from repetitions, belonging to earlier years. The modern character of the work is shown, for instance, in the articles "United States," "Presidential Campaign," "Theodore Roosevelt," "Aëronautic," "Naval Progress," "Philippine Islands," "Financial Review," and "Gifts and Bequests"—titles which I have selected almost at random. The treatment of topics seems to be careful, accurate, and non-partisan.

Everybody feels the expensive necessity of buying new books in order to keep up with the times; everybody resents that necessity, when it involves the purchase of a new edition of an old book, simply on account of a small proportion of new material which has been added to it. Consequently, I think, everybody should welcome a book which is practically all new, as well as useful.—R. W. R.]

IOWA—STATE GEOLOGIST. Annual Report, 1st and 2d. Des Moines, F. W. Palmer, 1868. (Purchase.)

IRON AGE DIRECTORY, 1909. New York, David Williams Co., 1909. (Gift of Publishers.)

IRRIGATION IN SOUTH DAKOTA. (Bulletin No. 210, U. S. Department of Agriculture, Office of Experiment Stations.) By S. H. Lea. Washington, U. S. Government, 1909. (Gift of U. S. Experiment Station.)

IRRIGATION IN THE SACRAMENTO VALLEY, CALIFORNIA. (Bulletin No. 207, U. S. Department of Agriculture, Office of Experiment Stations.) By S. Fortier. Washington, U. S. Government, 1909. (Gift of U. S. Experiment Station.)

LIST OF PERMISSIBLE EXPLOSIVES TESTED PRIOR TO MAY 15, 1909. (Explosive Circular No. 1, U. S. Geological Survey.) N. p., n, d. (Gift.)

LOCK CANAL AT PANAMA. By John R. Freeman. (House Document No. 10, Pt. 3.) Washington, U. S. Government, 1909. (Gift.)

MACHINE TOOL TRADE IN GERMANY, FRANCE, SWITZERLAND, ITALY, AND UNITED KINGDOM. By G. L. Carden. Washington, U. S. Government, 1909. (Gift of U. S. Bureau of Manufactures.)

MAGNETIC OBSERVATORY AT BALDWIN, KANSAS, 1901–1904. Washington, U. S. Government, 1909. (Exchange.)

MATERIALIEN ZUR GEOLOGIE RUSSLANDS. Band 23, Pt. 2. St. Petersburg, 1908. (Exchange.)

MINERAL RESOURCES OF THE ROCK ISLAND-FRISCO LINES, 1908. (Map.) (Gift.)

NEW YORK STATE WATER-SUPPLY COMMISSION. Annual Report, 2d and 4th. Albany, J. B. Lyon Co., 1907, 1909. (Gift of State Water-Supply Commission of New York.)

NEW YORK STATE WATER-SUPPLY COMMISSION. Studies of Water Storage for Flood Prevention and Power Development in New York State under Public Ownership and Control. Albany, J. B. Lyon Co., 1908. (Gift of State Water-Supply Commission of New York.)

ONTARIO BUREAU OF MINES. Annual Report, 16th, 1907. Toronto, L. K. Cameron, 1907. (Exchange.)

PEAT DEPOSITS OF MAINE. (Bulletin No. 376, U. S. Geological Survey.) By E. S. Bastin and C. A. Davis. Washington, U. S. Government, 1909. (Exchange.)

PILOT CHART OF THE NORTH ATLANTIC OCEAN. May, 1909. (Gift of U. S. Hydrographic Office.)

PILOT CHART OF THE SOUTH ATLANTIC OCEAN. June, July, and August, 1909. (Gift of U. S. Hydrographic Office.)

PRECISE LEVELING IN THE UNITED STATES, 1903-1907. Washington, U. S. Government, 1909. (Exchange.)

PRESERVATION OF IRON AND STEEL. (Bulletin No. 35, U. S. Department of Agriculture, Office of Public Roads.) By A. S. Cushman. Washington, U. S. Government, 1909. (Exchange.)

QUEBEC—COLONIZATION, MINES, AND FISHERIES DEPARTMENT. Mining Operations in the Province of Quebec, 1908. By J. Obalski. Quebec, C. Pageau, 1908. (Exchange.)

RAILWAY STATISTICS OF THE UNITED STATES OF AMERICA, 1908. Ed. 2. Chicago, R. R. Donnelley & Sons Co., 1909. (Gift of S. Thompson.)

REPORT ON THE GEOLOGICAL SURVEY OF CONNECTICUT. By C. U. Shepard. New Haven, B. L. Hamlen, 1837. (Purchase.)

REPORT ON THE PROPERTY OF THE MANHATTAN CONSOLIDATED GOLD MINES COMPANY OF NEVADA. By W. P. Jenney. Tonopah, 1909. (Gift of Author.)

RESULTS OF PURCHASING COAL UNDER GOVERNMENT SPECIFICATIONS. (Bulletin No. 378, U. S. Geological Survey.) By J. S. Burrows. Washington, U. S. Government, 1909. (Exchange.)

SHUSWAP SHEET, BRITISH COLUMBIA. Map, 1898. (Exchange.)

SURFACE WATER-SUPPLY OF NEBRASKA. (Water-Supply Paper No. 230, U. S. Geological Survey.) By J. C. Stevens. Washington, U. S. Government, 1909. (Exchange.)

U. S. AGRICULTURE DEPARTMENT. Report of the Secretary of Agriculture, 1908. (Abridged Edition.) Washington, U. S. Government, 1909. (Exchange.)

U. S. CIVIL SERVICE COMMISSION. Annual Report, 25th, 1908. Washington, U. S. Government, 1909. (Gift of U. S. Civil Service Commission.)

U. S. COMMISSIONER OF CORPORATIONS. Report of the Commissioner of Corporations on the Petroleum Industry. Pt. 1. Washington, U. S. Government, 1907. (Gift of U. S. Commissioner of Corporations.)

U. S. COMMISSIONER OF CORPORATIONS. Report of the Commissioner of Corporations on the Transportation of Petroleum, May 2, 1906. Washington, U. S. Government, 1906. (Gift of U. S. Commissioner of Corporations.)

U. S. HYDROGRAPHIC OFFICE. Hydrographic Bulletin No. 1029, May 19, 1909. (Gift of U. S. Hydrographic Office.)

U. S. HYDROGRAPHIC OFFICE. Notice to Mariners. No. 20, May 15, 1909. (Gift of U. S. Hydrographic Office.)

U. S. INTERIOR DEPARTMENT. Report of the Governor of Arizona to the Secretary of the Interior, 1908. Washington, U. S. Government, 1908. (Gift of U. S. Supply Division, Department of the Interior.)

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WESTERN AUSTRALIA GEOLOGICAL SURVEY. Bulletin No. 32. Perth, R. S. Sampson, 1908. (Exchange.)

WESTERN AUSTRALIA—MINES DEPARTMENT. Annual General Report of Western Australia, 1890. By H. P. Woodward. Perth, R. Pether, 1891. (Exchange.)

WISCONSIN—CONSERVATION COMMISSION. Report of the Conservation Commission of the State of Wisconsin, 1st. Madison, 1909. (Gift of Wisconsin Conservation Commission.)

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YEAR BOOK OF SCIENTIFIC AND LEARNED SOCIETIES OF GREAT BRITAIN AND IRELAND. Compiled from Official Sources. 25th annual issue. London, Griffin, 1908. (Purchase.)

GIFT OF MINISTERIO DE AGRICULTURE DE ARGENTINE REPUBLIC.

ANALES. SECCIÓN GEOLOGIA, MINERALOGIA Y MINERIA. Tomo I., Num. II., 3; tomo II., num. 1–3; tomo III., num. 2. Buenos Aires, 1905–1908.

APUNTES PARA LA CONFECCIÓN DE UN MAPA GEOLÓGICO AGRONÓMICO. Buenos Aires, 1906.

CATALOGO DE LA COLECCIÓN MINERALÓGICA PARA ENSEÑANZA SECUNDARIA. Buenos Aires, 1905.

CATALOGO DE LA COLECCIÓN MINERALÓGICA ESCOLAR PARA ENSEÑANZA PRIMARIA. Buenos Aires, 1905.

CATÁLOGO INSTRUCTIVO DE LAS COLECCIÓNES MINERALÓGICAS ESCOLARES. Buenos Aires, 1905.

INFORME SOBRE UNA EXPLORACIÓN GEOLÓGICA EN LA REGIÓN DE ORÁN. Buenos Aires, 1906.

INSTRUCCIONES PARA LA RECOLECCIÓN DE MUESTRAS DE ROCAS YACIMIENTOS METALIFEROS Y FÓSILES. Buenos Aires, 1905.

MEMORIA DE LA DIVISIÓN DE MINAS, GEOLOGIA É HIDROLOGIA, 1904–1905. Buenos Aires, 1905.

PLANO GENERAL DE LOS DISTRITOS MINEROS EXISTENTES EN LOS TERRITORIOS NACIONALES. N. d.

PRINCIPALES FENÓMENOS ORIGINADOS POR LOS TERREMOTOS Y MANERA DE OBSERVARLOS. Buenos Aires, 1908.

SEÑAS CONVENCIONALES PARA LAS SECCIONES DE LOS SONDEOS. Buenos Aires, 1906.

TRADE CATALOGUES.

N. E. GOODWIN & Co., Reno, Nev. "Rawhide Coalition" Special, showing market position of the company; returns of ores; mine- and mill-situation in Rawhide; ore-deposits and the vein-system on Coalition ground.

C. W. HUNT & Co., West New Brighton, N. Y. Catalogue No. 091, Coal and Ore-Handling Machinery, as hoisting- and conveying-machinery, elevators, cable-railways, coal-crackers, steam-shovels, wall-chutes, screens, blocks, and pulleys.

A. LESCHEN & SONS ROPE Co., St. Louis Mo. Photographs and description of an 8,000-ft. patent friction-grip wire-rope tramway over Lake Michigan, used to transport material excavated in the construction of the new water-supply tunnel for Chicago.

NEW YORK ENGINEERING Co., New York, N. Y. Empire hand prospecting-drill for mining- and prospecting-work. All-steel dredges for gold-dredging.

RENDALL CONSTRUCTION Co., 120 Liberty Street, New York, N. Y. The Rendall process for reducing ores at a smaller cost and greater facility than by the smelting-process. It is a direct process, consisting essentially in the roasting of ores with gas under pressure, eliminating the silica, and concentration prior to melting. The ores, except iron, are then melted on the ground and sent to the refiner.

STEPHENS-ADAMSON MFG. Co., Aurora, Ill. Conveying and Transmission; May, 1909, Vol. 4, No. 2. Containing cuts and descriptions of coal-handling machinery, wire screens, bucket- and other elevators, belt-conveyors, separators.

STROMBERG-CARLSON TELEPHONE MFG. Co., Rochester, N. Y. Bulletin No. 1000, September, 1908. Catalogue of various styles of mine-telephones, iron-clad magneto-telephones, magneto-switchboards, diagram of a metallic circuit for a mine phone-system.

WEIR BROS. & Co., 25 Broad St., New York, N. Y. A review of "Porphyritic Coppers, the present and future low-cost producers." Article gives the history, occurrence, cost of production, uses, and advantages of these coppers.

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members who accepted election during the month of June, 1909:

Members.

Adam A. Boyd, ... Broken Hill, N. S. W., Australia.
George Fairbairn, ... Celaya, Guanajuato, Mexico.
Vaughan M. Lavery, ... Goyllarisquisga, Peru, So. America.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed for election as members of the Institute during the month of June, 1909. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

John Barry, ... Animas Forks, Colo.
John Robert Finletter, ... Globe, Ariz.
Leigh Hill French, ... New Rochelle, N. Y.
Ole G. Hoaas, ... Wallace, Idaho.
Richard Horschitz, ... New York, N. Y.
Norman Lee Jenks, ... Butte, Mont.
Howard Waldo Kitson, ... New York, N. Y.
Leslie Montefiore Kozminsky, ... New York, N. Y.
Henry M. Lancaster, ... Wallace, Idaho.
Aquila Chauncy Nebeker, ... Topliff, Utah.
John Thomas Roberts, Jr., ... Tucson, Ariz.
Joseph Henry Rodgers, ... Seattle, Wash.
Michael J. Slattery, ... San Martin Hidalgo, Jalisco, Mex.
James Burnett Torbert, ... Cerro de Pasco, Peru, So. America.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of June, 1909. This list, together with the lists given in the *Bulletin* for February, March, April, May, and June, therefore, supplements the annual list of members corrected to Jan. 1, 1909, and brings it up to the date of July 1, 1909. The names of Members who have accepted election during the month (new members), are printed in *italics*.

ADAMS, ARTHUR K., Mineral Inspector.........General Land Office, Huron, S. D.
ALLEN, ROBERT, Met. Engr......................Apartado 2554, Mexico City, Mexico.
ANDERSON, WILLIAM G., El Favor Mining Co.....Hostotipaquillo, Jal., Mexico.
ARNOLD, C. E. LE N., Veteran Mine.. ....Kimberly, Nev.
BALL, SYDNEY H., Care F. H. Ball.................Title & Trust Bldg, Chicago, Ill.
BANON, HENRY C., P. O. Box 144....Chemainus, Vancouver Isd., B. C., Canada.
BARROWS, WALTER A., JR..........................14 Villa Beach, Collinwood, Ohio.
BATES, MOWRY, Care W. R. Holligan & Co......111 Broadway, New York, N. Y.
BAXTER, FRANCIS K., JR...................423 Wells Fargo Bldg., San Francisco, Cal.
BLUMENAU, HERMAN...........Landhausstrasse 48, Berlin-Wilmersdorf, Germany.
**Boyd, Adam A.*, Mine Mgr., Proprietary Mine, Broken Hill, N. S. W., Australia. '09.
BROWN, A. T., Cons. Min. Engr., Broken Hill Chambers, 31 Queen St., Melbourne, Vic., Australia.
BUELL, LLOYD T., Ray Central Copper Mining Co...... ....Ray, Ariz.
CARR, HENRY C............................Apt. 51, The Wyoming, Washington, D. C.
CHAMBERS, FRANK M., Supt., Western Ore Purchasing Co., P. O. Box 1752, Goldfield, Nev.
CHASE, FRANK D., Compania Minera Fundidora y, Afinadora "Monterey," Apartado 27, Monterey, Mexico.
CLEMENTS, J. MORGAN, Cons. Min. Geol. and Engr., 42 Broadway, New York, N. Y.
COE, IRA J..883 Flood Bldg., San Francisco, Cal.
COLBATH, HARRY.................................P. O. Box 717, Salt Lake City, Utah.
CRAWFORD, HENRY E.............123 Bay 29th St., Bensonhurst, Brooklyn, N. Y.
CREMER, FELIX, Butters Copala Mines, Inc.................Copala, Sinaloa, Mexico.
DANIEL, J. M., JR., La Mina Leonora..... ....Calvillo, Aguas., Mexico.
DIXON, JAMES T............................Salairsky Rudnik, Govt. of Tomsk, Russia.
DOUGLASS, ROSS E., Care Thomas Bros. & Metcalf, 516 Grant Bldg., Los Angeles, Cal.
DUNSTON, ALFRED J., Min. Engr., Kyloe Copper Mines, Adaminaby, N. S. W., Australia.
DURFEE, ELMER W., Mgr., Alvarado Gold Min. Co............Congress Jc., Ariz.
ELLAN, A. SPENCER, 16 Arlington Pk. Mansions, Chiswick, W., London, England.
**Fairbairn, George*, Min. Engr...Apartado 31, Celaya, Guanajuato, Mexico. '09.
FOSTER, GEORGE C...521 E. 18th St., Denver, Colo.
FRANCK, ROBERT P...Box 452, Imperial, Cal.
GARTNELL, HERBERT W., School of Mines..............Kalgoorlie, West Australia.
GIFFORD, ALVAH W.........................Apartado 47, Monterey, N. L., Mexico.

GILKERSON, HENRY G.....................318 Judge Bldg., Salt Lake City, Utah.
GRAVE, PERCY, Supt., Suriana Min. & Smltg. Co., Achotla,
Campo Morado, Guerrero, Mexico.
HAMILTON, S. HARBERT, Min. Geol., Hamilton & Hansell, 29 Broadway,
New York, N. Y.
HENDERSON, ENOCH..Wickenburg, Ariz.
HEISE, A. ROY..1253 Jones St., San Francisco, Cal.
HERR, IRVING..47 Prentiss St., Cambridge, Mass.
HOLDEN, EDWIN C....................................15 Myrtle St., White Plains, N. Y.
HOLLOWAY, WILFRID S...Sakju, Korea.
HOLT, M. B...2 W. 94th St., New York, N. Y.
INGERSOL, JOHN W., Chem., Continental Copper Mining & Smelting Co.,
Hill City, S. D.
JOHNSON, OWEN...211 Spicer Ave., Victor, Colo.
*_King, Charles F._, Bureau of Buildings, Tremont and 3d Aves.,
New York, N. Y. '08.
KING, HAROLD F., Abangarez Gold Fields of C. R., Ltd., Puntarenas,
Costa Rica, C. A.
*_Lavery, Vaughan M._, Min. Engr.......Goyllarisquisga, Peru, So. America. '09.
LAWRENCE, WILLIS, Genl. Supt., Sierra Morena Mining & Refining Co.,
Paso Robles, Cal.
LEVINGS, G. VAN B....................................Apartado 22, Parral, Chih., Mexico.
LEWIS, HENRY M., JR..P. O. Box 93, Garfield, Utah.
McCAUSTLAND, ELMER J.........................5264 19th Ave., N. E., Seattle, Wash.
McCOLLOM, CHARLES R., Supt., Torreroca Min. Co................Steeplerock, N. M.
MACKAY, ANGUS R..Wickenburg, Ariz.
MARSH, RICHARD...304 Post St., Spokane, Wash.
MAXWELL, WILLIAM...........Parliament House, Brisbane, Queensland, Australia.
MAYER, PAUL H...Instructed to hold all mail.
MORSE, WILLARD V...1732 Wazee St., Denver, Colo.
MORTON, ERLE D...Box 133, Hollywood, Cal.
MYERS, D. B., Min. Engr..............380 Pacific Electric Bldg., Los Angeles, Cal.
NAKAMURA, KEIJIRO, Sumitomo's Smltg. Wks.,
Shisaka-Shima, Ochi-Gun, Iyo, Japan.
NIEDING, BURTON B., San Toy Mining Co..............Sta. Eulalia, Chih., Mexico.
PALSGROVE, HARRY G., Vindicator Mine..........................Independence, Colo.
PEARCE, RICHARD, Care G. B. Pearce......................Hayle, Cornwall, England.
PAZ-SOLDAN, FRANCISCO A.....Miraflores-Alameda 13, Lima, Peru, So. America.
PERKS, HARRY B., Mgr., Florence Placer Mining Co.................Adams, Idaho.
RAYMOND, R. M...Apartado 740, Mexico City, Mexico.
REID, ROBERT S...............................Calle Jofre 422, Santiago, Chile, So. Am.
RICHMOND, A. B..227 So. 4th Ave., Tucson, Ariz.
ROBBINS, HALLET R., Colonial Gold Mining Co........Keremeous, B. C., Canada.
ROBERTSON, JAMES D...Apartado 79, Durango, Mexico.
ROBINS, THOMAS, JR...72 Front St., New York, N. Y.
ROPP, ALFRED VON DER.................................Alaska Bldg., San Francisco, Cal.
SAKIKAWA, MOTARO.......................1116 Omachi, Kamakura, Sagami, Japan.
SCHMIEDELL, JOHANN H., Care Aschoff & Co., Market Bldgs.,
Mincing Lane, London, E. C., England.
SCHORR, ROBERT...........................Postal Telegraph Bldg., San Francisco, Cal.
SEAL, ALBERT E..................8 The Grove, Muswell Hill, London, N., England.
SLATER, AMOS, Min. Engr.......................American Bank Bldg., Seattle, Wash.

SMITH, LLOYD B...R. F. D. 36, Laceyville, Pa.
SMITH, S. K., Genl. Mgr., Vinton Colliery Co..........................Vintondale, Pa.
STEWART, HOWARD R., State Dept. of Alajoas,
Maceio (via Southampton), Brazil, So. Amer.
SULLY, JOHN M..Santa Rita, N. M.
TOM, ISIDORE, Gt. Boulder Perseverance Gold Mine..Kalgoorlie, West Australia.
TRUMBULL, LOYAL W..Van Vleck, Texas.
TURNER, R. CHESTER.................................2511 Bancroft Way, Berkeley, Cal.
VAIL, HERBERT E., Mgr., Hannans Star, Ltd.....................Kamballie, Australia.
VAN LAW, CARLOS W...............................165 Broadway, New York, N. Y.
VENABLES, H. L., Anglo-So. American Bank.........Antofagasta, Chile, So. Amer.
WADSWORTH, M. E., School of Mines, Univ. of Pittsburg, Grant Blvd.,
Pittsburg, Pa.
WAGONER, LUTHER, Cons. Engr..................850 Union St., San Francisco, Cal.
WARD, OSMER B., Lake View Consols, Ltd., P. O. Box 99, Finiston, West Australia.
WEAVER, ISAAC S., Care E. Harmon..............715 Reservoir St., Baltimore, Md.
WILMOT, H. CLIFFORD, Min. Engr.........Newhouse Bldg., Salt Lake City, Utah.
WOOD, LEE S..1524 Detroit St., Denver, Colo.
WOODBURY, L. S ....P. O. Box 249, Placerville, Cal.
WORTH, JOHN G..Tonapah, Nev.
ZERENER, MARTIN B.................................165 Broadway, New York, N. Y.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address on Records, from which Mail has been Returned.
Adams, Randolph,	Copperhill, Tenn.
Alexander, George E.,	Sparta, Ore.
Andrew, Thomas,	Pretoria, So. Africa.
Arozarena, R. M. de,	Mexico City, Mexico.
Bassett, Thomas B.,	Cumpas, Sonora, Mexico.
Batchelder, Joseph F.,	54 1st St., Portland, Ore.
Bellam, Henry L.,	Reno, Nev.
Brook, Henry E. C.,	Cadia, N. S. W., Australia.
Brown, Frank H.,	Coppermount, Alaska.
Campa, Jose,	Mexico City, Mexico.
Collins, W. J.,	Johannesburg, So. Africa.
Cragoe, A. Spencer,	Vencedora, Mexico.
Dougherty, Clarence E.,	41 Wall St., New York, N. Y.
Ekberg, Benjamin P.,	Johannesburg, Transvaal, So. Africa.
Field, Wilfrid B.,	Mexico City, Mexico.
Fitzsimmons, F. J.,	Cananea, Mexico.
Francis, George G.,	177 St. George's Sq., London, W., England.
Fuller, Frederick D.,	Sumpter, Ore.
Gage, Edward C.,	San Dimas, Dur., Mex.
Gee, Emerson,	Reno, Nev.
Hollister, John J.,	Mexico City, Mexico.
Hunt, Thatcher R.,	Iron Mt., via Keswick, Cal.
Jackson, Byron N.,	Milton, Cal.
Jessop, Herbert J.,	Guanacevi, Mexico.
Jewett, Eliot C.,	2918 Morgan St., St. Louis, Mo.
Judd, Henry A.,	Mertondale, W. Australia.
Kow, Tong Sing,	Shanghai, China.

Mildon, Reginald B.,	Nacozari, Son., Mexico.
Moulton, Herbert G.,	Cobalt, Ont., Can.
Muir, Thomas K.,	Portland, Ore.
Nawatny, William F.,	Harrisburg, Ill.
Philbrick, Arthur,	Manhattan, Nev.
Piper, John W. H.,	Buenos Ayres, Argentine Rep., S. A.
Potter, J. A.,	41 W. 124th St., New York, N. Y.
Reynolds, Llewellyn,	Guanajuato, Mexico.
Rigney, Thomas P.,	Reno, Nev.
Rodda, Richard W.,	Seattle, Wash.
Sandifer, Harmer C.,	El Oro, Mexico.
Schlemm, William H.,	Durango, Mexico.
Scott, Winfield G.,	Long Beach, Cal.
Skelding, Joseph F.,	Embreeville, Tenn.
Thomas, Richard A.,	43 Wall St., New York, N. Y.
Vaux, Charles A.,	P. O. Box 80, East Rand, So. Africa.
Warren, Henry L. J.,	Salt Lake City, Utah.
Wolfe, Burton L.,	Ely, Nev.
Young, William,	Kenora, Ont., Canada.

NECROLOGY.

The deaths of the following members have been reported to the Secretary's office during the month of June, 1909:

Date of Election.	Name.	Date of Decease.
1878.	*L. G. Laureau,	June 6, 1909.
1903.	*Frank A. Lucy,	June 7, 1909.
1887.	*Charles C. Mattes,	June 8, 1909.
1903.	*Delos V. A. Williams,	May 15, 1909.

BIOGRAPHICAL NOTICES.

Louis G. Laureau was born in France about 1841, and came to this country at the time of the War of the Rebellion. He became, and remained for many years, the chief assistant of Alexander L. Holley, whose work on ordnance and armor, and on the Bessemer and open-hearth steel processes and apparatus, was greatly aided by Mr. Laureau's technical knowledge and skill. After Holley's death, he practiced as a consulting, mechanical, and metallurgical engineer and expert in patent causes, and was for some time a member of the firm of Gordon, Strobel & Laureau, consulting, designing, and constructing metallurgical engineers.

Mr. Laureau joined the Institute in 1878, and contributed to its *Transactions* in 1885 a paper on A Bessemer Converter-House without a Casting-Pit (*Trans.*, xiii., 697). He died at his home in Yonkers, N. Y., June 6, 1909.

* Member.

Frank Allen Lucy, born Jan. 13, 1877, in Denver, Colo., and graduated as a mining engineer from the Colorado State School of Mines at Golden, was employed in 1903 as surveyor and mining engineer by Stevens, Barbour & Co., Idaho Springs, Colo.; became, about the end of 1904, U. S. deputy mineral surveyor at Goldfield, Nev., and was appointed superintendent of the Florence mine at that place in May, 1909. On June 7, a few days after his appointment, he met his death by falling down the main shaft of the mine. Mr. Lucy became a member of the Institute in 1903, and had made himself creditably known in the mining-districts of Nevada, when he was thus tragically cut off in a most promising career, just before his 32d birthday.

Delos Van Alstyne Williams was born Sept. 23, 1854, at Syracuse, N. Y.; subsequently lived at Joliet, Ill.; studied at Chicago University; went to Leadville, Colo., in 1878; and acted at various times thereafter as superintendent of mines at Leadville and other points in Colorado. Subsequently removing to Mexico, he took charge of the San Francisco del Oro, in Parral (1891); the Garbatos, in Durango (1893 and '4); the Mecalina, in Parral (1899); and La Providencia (1902). He was elected a member of the Institute in 1903, at which time he was connected with several Mexican mining companies. In 1904, he made his headquarters at the City of Mexico, as one of the firm of Williams & Yandle; and in later years he had to do as mining engineer and manager with mines at Alameda and in Zacatecas, including the mine of La Reforma company, Valenciana. In 1908, he became owner of the Sidney Group at Johannesburg, near Randsburg, Cal., which he personally managed. In January, 1909, Mr. Williams was lost in Death Valley, and went eight days without food and three days and a half without water—during which period he walked 200 miles, while numerous parties in automobiles and on horseback were searching for him. It was not supposed that this experience of exertion and suffering had seriously impaired his health; but he was somewhat suddenly attacked, not long after, by the illness of which he died, after an operation, May 15, 1909, at the California hospital, Los Angeles, Cal. He was buried at Joliet, Ill., the home of his youth. His widow and son survive him.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Formation and Enrichment of Ore-Bearing Veins.

SUPPLEMENTARY PAPER.

BY GEORGE J. BANCROFT, DENVER, COLO.

(Spokane Meeting, September, 1909.)

At the New York meeting of the Institute (April, 1907), I presented a paper entitled, The Formation and Enrichment of Ore-Bearing Veins,[1] in which paper I advanced the following propositions:

(1) That the majority of mineralized veins are the product of expiring vulcanism; (2) that most of these veins were primarily mineralized by comparatively rich solutions in comparatively short periods of time; (3) that the solutions gained their metal-values from a comparatively rich source; (4) that there is a barysphere containing large amounts of the useful metals; (5) that eruptions spring from various depths and bring various kinds of magmas towards the surface; and (6) that only those eruptions which disturb the barysphere and bring a magma rich in metals sufficiently near the surface to be leached by vein-making solutions are productive of valuable ore-deposits, other eruptions producing barren veins. Ore-deposits due to magmatic segregation were not included in this general survey.

As a result of considerable further study I have modified my views in some respects, while in others I feel more sure than ever of the ground taken at that time.

That ore-bodies are infrequent occurrences and born of extraordinary conditions I think is now generally accepted. The theory that persisted for a time—namely, that ore-bodies were formed by the ordinary ground-water, which consists of extremely dilute solutions, derived from leaching extremely lean surface-rocks, and which must occupy enormously long periods of time in concentrating the values so leached, is, I think, now pretty generally regarded as not applicable to the great majority

[1] *Trans.*, xxxviii., 245 (1908).

of our mining-districts, although it may account for a few isolated deposits.

I note that a few recent writers still use this theory as a sort of "point of departure" for their discussions, but the majority seem to realize that the tendency of ordinary ground-water circulation is to diffuse any soluble matter rather than to concentrate it, and that an unusual precipitant must be present to provoke an important concentration under this hypothesis.

That the forces of expiring vulcanism are the agencies which account most logically for ore-bodies is an opinion very generally held, not only because of the intimate association of ore-bodies with eruptive rocks, but also because the study of active volcanoes and of the springs rising near them shows that ore-making agencies to a limited extent, at least, are at work there. Thus, A. Lacroix[2] found pyrite, pyrrhotite, and galena, together with sulphates of sodium, potassium, calcium, magnesium, and aluminum, in the sublimates of fumaroles on Vesuvius; J. W. Mallet[3] discovered silver in volcanic ash at Cotopaxi and Tunguragua; O. Silvestri[4] found copper in the fumes of Etna, while traces of practically all the common metals have been found in eruptive rocks.

Similarly, it has been recognized that the period of expiring vulcanism could not have been of very long duration, geologically speaking, although it has been shown that in some districts the repeated recurrence of eruptive action has had the effect of continuing the mineralizing action through long periods of time.

My fifth and sixth propositions have not been so well established. I think it is generally admitted that certain eruptive rocks produce mineralized areas, while others do not; this suggests most forcibly that the eruptives themselves differed very decidedly in the matter of mineral contents; and it seems reasonable to infer that those eruptives which do produce mineral richness have been rich themselves and probably originated at great depth. The advance of science has still further substan-

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[2] *Bulletin de la Société Française de Minéralogie*, vol. xxx., p. 219 (1907).

[3] *Proceedings of the Royal Society*, vol. xlii., p. 1 (1887); vol. xlvii., p. 277 (1889–90).

[4] *I Fenomeni Vulcanici presentati dell' Etna, etc.* (Catania, 1867), per *Bulletin No.* 330, *U. S. Geological Survey*, p. 217 (1908).

tiated the theory that the earth has a very heavy center, and it is reasonable to suppose that this increased specific gravity is partly due to relatively large metal-contents. It is not, however, generally conceded that it is necessary for a mineralizing eruptive to be so rich in metals and so heavy that it would rarely or never reach the surface, but would form laccolites, according to my old hypothesis. Nor do I longer hold this view. At that time I remarked:

"If it could be shown that the surface eruptive rocks have a tendency to throw off metals, as they do steam and sulphur, during the cooling process this would remove many of my objections to considering them the source of the metals in our ore-bodies. In the lack of such proof, however, we must recognize that they are extremely lean, and therefore a very unlikely source of mineral wealth."

Now, I think there are very good reasons for believing that this very thing is true—namely, that eruptive rocks do have a tendency to throw off their metal-contents during the cooling process, or rather as soon as they reach a horizon of lessened pressure, which condition is apt to be coincident with cooling.

The wonderful crystallographic researches of J. E. Spurr, Waldemar Lindgren, and others, have shown that magmas are totally different from dry melts, and the cooling of a magma is accompanied by a remarkable series of differentiations. Mr. Spurr has shown how the metals would be concentrated either in a very base or a very acid magma, and finally, in the case of the acid magma, would be extruded together with pure silica and water, thereby forming veins. I think, however, that those who have made a great deal of the powerful agency of magmatic differentiation have overlooked a very active contemporary agent—namely, chlorine. Bromine, iodine, and fluorine may be equally active agencies in volcanic emanations; but as these elements are relatively rare, I shall confine myself here to the very active part which chlorine may play in carrying away from the hot eruptive its metal-contents and depositing the same in ore-bodies. These considerations have been suggested to me by studying the dry chlorination-process for the treatment of complex ores, as developed by J. L. Malm, at Corbin, Mont., and at Denver, Colo. This process depends primarily upon the facts that, in the dry state, chlorine has a greater affinity for the metals than sulphur or oxygen, and that the chlorides of all the metals are soluble together in hot water.

Thus, cupric chloride, lead chloride, zinc chloride, gold chloride, and iron chloride are soluble in hot water direct, and silver chloride is soluble in hot cupric chloride.

Now, it is noticeable that chlorine is nearly always present in volcanic emanations. Thus, J. W. Judd says:

"The most abundant of the substances which are ejected from volcanoes is steam or water-gas, which, as we have seen, issues in prodigious quantities during every eruption. But with the steam a great number of other volatile materials frequently make their appearance. The chief among these are the acid gases known as hydrochloric acid, sulphurous acid, sulphuretted hydrogen, carbonic acid, and boracic acid; and with these acid gases there issue hydrogen, nitrogen, ammonia, the volatile metals arsenic, antimony, and mercury, and some other substances."[5]

After dwelling upon the large amount of CO_2 present in volcanic gases, Chamberlin and Salisbury have the following to say with reference to emanations:

"Sulphur gases are very common accompaniments of volcanic eruptions. They take the forms of sulphuretted hydrogen and sulphurous acid and perhaps of sublimated sulphur, all of which are liable to pass by oxidation and hydration into sulphuric acid. Chlorine and hydrochloric gases are also common, particularly at high temperatures. Fluorine and other gases are occasionally present."[6]

T. Wolf found that near the crater of Cotopaxi the fumes were mostly of hydrochloric acid with some free chlorine, while at lower levels hydrogen sulphide was found with a trace of sulphur dioxide.[7]

Sainte-Claire Deville found chlorides of iron and copper in a fumarole of Vesuvius.[8]

R. Bunsen found various metallic chlorides in the sublimates around fumaroles of Mt. Hecla in Iceland.[9]

A characteristic of volcanic emanations is, that chlorine is found either free or as the chloride of the metals, or as hydrochloric acid, while a characteristic of hot mineral springs is that the water contains large quantities of the chlorides of sodium and potassium. Granting that such hot mineral springs

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[5] *Volcanoes: What They Are and What They Teach*, p. 40 (1881).

[6] *Geology*, 2d ed., vol. i., p. 619 (1906).

[7] *Neues Jahrbuch für Mineralogie, Geologie und Palaeontologie*, p. 163 (1878).

[8] *Bulletin de la Société Géologique de France*, Second Series, vol. xiii., p. 606 (1855–56).

[9] *Annales de Chimie et de Physique*, Third Series, vol. xxxviii., p. 215 (1853).

are related to eruptions, and that the chlorine in both emanations and springs had a common origin, it is evident that the part which has escaped by the medium of spring-water has undergone certain reactions, which the part which escaped as a hot gas, from an open vent, did not undergo. Does not this suggest what may take place in case the chlorides are extruded through crevices or veins where they may encounter gradually lessened temperatures together with water?

Now, it is known that the temperature of fluid magmas may range from 2,000° to 3,000° F.[10]

Let us suppose that, under a temperature of 1,000° C., we had a magma which contained chlorine, water-gas, sulphur, silver, copper, lead, iron, zinc, and gold. Of course, there would also be other elements present, but, as previously stated, I shall not try to cover the whole field. Let us suppose that there is sufficient chlorine to form chlorides with all the metals and an excess besides. Any hydrogen present would be combined with the chlorine, the affinity for chlorine being greater than for sulphur. Now, let us try to imagine what would happen as this magma approached the surface.

At the temperature given, the chlorine would attack all the metals, except gold, and the chlorides would all be gases, for it is well known that the metallic chlorides are all volatile at relatively low temperatures.

The chloride of gold under atmospheric conditions decomposes at about 120° C. Of course, the volcanic gases are under some pressure even when escaping from the magma, and, as pressure raises the temperature of decomposition, it is difficult to say just how cool the magma must become before the gold would accompany the other metals.

The fact that gold is often found by itself in a state of great purity may be accounted for by its isolation from the other metals as regards chemical reactions. Thus, at Farcum Hill, Breckenridge, are found deposits of most beautiful crystalline gold by itself, while less than a mile away are large deposits of complex ores. The complex ores occur in the eruptive dikes or on the contacts, while the gold is found in carbonaceous shale. It is well known that carbon will precipitate gold from

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[10] *Geology*, Chamberlin and Salisbury, 2d ed., vol. i., p. 615 (1906).

a chloride solution, as is done in wet-chlorination mills, but it will not precipitate the other chlorides.

The chlorine would not attack the silicates of the eruptive, as is shown by the experiments of Brun,[11] who heated a Lipari lava and observed the following exhalations:

From 0° to 825°, volatilization of water.

At 825°, first evolution of chloride vapors.

From 874° to 1,100°, temperature of explosions.

At 1,100°, mean temperature of flowing lava.

Although the dry-chlorination process does not use high temperatures, Mr. Malm has experimented with chlorine and the ordinary rock-minerals at high temperatures and shown that they do not react.

The sulphur would occur as sulphur gas and partly as sulphur chloride. There would, of course, be a great deal of water-gas present. As the magma rose to a horizon of lessened pressure these gases would expand and leave the magma, bursting out through any veins or vents or porous strata that might provide a means of escape. The great distance to which the metals have penetrated porous strata, as, for instance, at Morenci, Ariz., may be accounted for in this way.

The farther from the magma the gases traveled the cooler they would become, and as they became cooler they would become a liquid. We would then have the chlorides of the metals in a hot aqueous solution, together with sulphur chloride and elemental sulphur, the latter in an extremely fine state of subdivision (as it is found in many mineral springs).

Precipitation would not take place till some precipitating-agent was encountered in the rocks.

The cooling of the solution would throw down the lead and the lead only. Silver chloride is difficultly soluble and easily precipitated. This may account for the exceeding purity of some lead-deposits and the frequent association of silver and lead.

If CaO in abundance were encountered all the metals would be precipitated together. This may account for the complex nature of many deposits in porphyry-lime contacts, as, for instance, at Leadville, Rico, Breckenridge, etc.

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[11] *Archives des Sciences Physiques et Naturelles*, Fourth Series, vol. xix., pp. 439, 589 (1905).

It is fair to assume that the heated eruptive would drive off the CO_2 from any limestone in immediate contact with it. The chloride gases emanating from the same eruptive would find an immediate and abundant precipitant in the form of CaO thus formed, or, in case conditions became cool enough for water to convert the oxide to the hydroxide, the latter would be equally efficient as a precipitant. Thus,

$$2\ MCl_x + xCaO = MO_x + M + xCaCl_2$$
$$2\ MCl_x + xCaO + xH_2O = 2\ M\ (OH_x) + xCaCl_2.$$

In granitic or eruptive rocks, the methods of precipitation are not quite so simple. I have immersed pieces of Boulder county granite in solutions of the metallic chlorides, sulphur chloride, and a little free acid, and at the end of three days there was no appreciable result. Again, I have boiled pieces of the same granite in strong hydrochloric and sulphuric acid, and in half an hour the reactions were very considerable. The hydrochloric acid was the more active. The piece of granite was eaten away considerably. Examination of the remaining piece with a magnifying-glass showed that the surface was reduced to a covering of spongy silica and white mica. Whether the mica had been whitened by leaching the iron, or whether only those bits remained intact which had no iron, I am not prepared to say. In addition to the piece remaining undissolved (at the end of 30 min.), there was a sediment which, on examination, appeared to be rounded and porous pieces of silica and minute flakes of pure white mica. The test with sulphuric acid rendered the piece darker. Only a very small part went into solution. On examining with the glass, the feldspars were lacking and the surface was covered with a coating of pearly silica and black mica. In this connection Clarke says:

"Hot waters, charged with sulphuric or hydrochloric acid, attack nearly all eruptive rocks, dissolve nearly all bases, and leave behind, in many cases, mere skeletons of silica." [12]

On this subject Judd says:

"In many volcanoes the constant passage through the rocks of the various acid

[12] *Bulletin No.* 330, *U. S. Geological Survey*, p. 408 (1908).

gases has caused nearly the whole of the iron, lime, and alkaline materials of the rocks to be converted into soluble compounds known as sulphates, chlorides, carbonates, and borates; and, on the removal of these by the rain, there remains a white, powdery substance, resembling chalk in outward appearance, but composed of almost pure silica. There are certain cases in which travelers have visited volcanic islands where chemical action of this kind has gone on to such an extent, that they have been led to describe the islands as composed entirely of chalk."[13]

F. Henrich shares the same view, claiming that the chlorides of potassium and sodium found in sublimates of aqueous fumaroles were formed by the action of moisture and hydrochloric acid on the alkaline silicates of the heated lavas.[14]

We know from observation underground that granitic and other rocks are attacked by mineral solutions which produce first a softening of the rocks, and eventually remove all the original constituents except the silica and hydrous alumina silicates.

It is, of course, difficult to reproduce in the laboratory with a few simple salts the complex reactions that take place far underground; and here at least we must take the evidence as we find it, even if it is difficult to reproduce the reactions.

We find chlorine or hydrochloric acid or metallic chlorides issuing from volcanoes, and we find sodium, calcium, and potassium chlorides issuing from mineral springs; so it seems highly probable that the sodium and potassium silicates break up, and the chlorine, after it has become cooled below the boiling-point of water, reacts with the sodium or potassium, dropping any metals with which it may have been combined.

The reactions leading to the formation of calcium chloride have been explained above. The reactions with the silicates would most naturally begin with the free hydrochloric acid, thereby liberating hydrogen, which would combine with the sulphur to form H_2S, and this in turn would precipitate the metals according to the following reactions:

$$MCl_x + \frac{x}{2}H_2S = xHCl + MS_{\frac{x}{2}},$$

and would convert any metals already precipitated by other agencies to the sulphides, as follows:

[13] *Volcanoes: What They Are and What They Teach*, p. 41 (1881).

[14] *Zeitschrift für angewandte Chemie*, vol. xix., No. 30, p. 1326 (July 27, 1906); vol. xx., No. 5, p. 179 (Feb. 1, 1907).

$$MO_x + xH_2S = MS_x + xH_2O$$
$$M + xH_2S = MS_x + xH_2.$$

The first reaction would result in more hydrochloric acid, which would be available to repeat the cycle.

Whether the metals are precipitated by CaO or by reactions with the silicates, they would be almost at once converted to sulphides, according to the above reactions; and that is the condition in which we find them. Gold, which is precipitated by carbon, would not be subject to the action of H_2S, and hence would likely occur as free gold in carbonaceous formations, which agrees with the facts as we find them.

As stated above, I have confined myself, for the sake of simplicity, to an arbitrarily-chosen condition. It is my purpose to point out the very active part which the halogens may play in divesting an eruptive of its metal-contents, and conveying the same into neighboring veins or openings in the rocks, rather than to prescribe the exact steps that are followed. It is conceivable that under certain conditions the above-mentioned reactions may be very important, while in others they may be negligible. It is conceivable that in some cases an eruptive may be totally divested of its useful metals and yet little of it be precipitated short of the atmosphere. This may account for the Snake river placer gold, which is extremely fine, and so often found associated with volcanic ash as to provoke the theory that it had an atmospheric origin. On the other hand, a metalliferous eruptive may be only partly relieved of its metal-contents, the residue remaining in a more or less segregated form in the body of the eruptive, as at Ely, Nev., where the sedimentaries adjoining the eruptive area contain veins and deposits of copper-ore, while the eruptive itself contains large masses of low-grade ore which appear to be due to magmatic segregation.

I think these considerations, helping to explain some of the puzzling things about ore-bodies, may be of service in promoting progress towards the goal of a perfect understanding of the subject.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Review of Modern Cyanide Practice in United States and Mexico.

BY S. F. SHAW, LOS ANGELES, CAL.

(Spokane Meeting, September, 1909.)

CONTENTS.

THIS paper is a review of the principal details of cyanide practice in several of the modern plants in America, mainly during the year 1908. Two of the mills, the Goldfield Consolidated and the Veta Colorado, are not yet prepared to give data, the former having started about Jan. 1, 1909, and the latter not yet being completed. These two mills embody most of the latest methods and are therefore described as far as possible.

I. CRUSHING.

The principal methods of crushing ores for treatment by the cyanide process are with gyratory and jaw-crushers, stamps, Bryan and Huntington mills, and tube-mills. The first stage is usually done by gyratory or jaw-crushers, in a few cases sup-

plemented by rolls, followed either by stamps, or by Bryan or Huntington mills, and, if very fine comminution be necessary, completed by tube-mills.

1. *Crushers.*—Owing to the simplicity of operation and low cost of wearing-parts, jaw-crushers, usually of the Blake type, are more generally used than gyratory crushers, although the latter are being installed in many modern mills, while indications point to a possible complete replacement. The Dodge crusher seems to find little favor, and is used in only one of the mills described in this paper.

The material delivered by a gyratory crusher, being more uniform in size, provides a better product for either the stamps or the Huntington mills. The wear of metal on any crusher is low, seldom exceeding 0.1 lb. per ton of ore handled, which points to the advisability of reducing the size to as fine a limit as practicable, which I consider to be nearer the size from 0.5 to 0.75 in. than from 1 to 1.5 in.

2. *Stamps.*—In most of the best mills, coarse crushing is followed by stamping, which, in view of the simplicity of machinery and low cost of labor, repairs, and renewals, appears to be a logical procedure. Stamps receive material from 0.5 to 1.5 in. in size, and crush it to from 12 to 50 mesh, with a duty of from 2.5 to 5 tons per stamp-head per 24 hr., depending upon the character of the ore, fineness of screens, and height of discharge.

Stamping tends to make slime, and when concentration only is used to save the precious metals, this slime is a serious drawback; but when the tailings are to be treated with cyanide, it is advantageous, especially since this fine material is so easily handled.

There is a marked tendency towards an increase in the weight of the stamp, which is now nearer 1,050 lb. than 850 lb., the practice several years ago. This tendency is even more marked in South Africa, where, in a mill of the latest design, an installation of stamps each weighing 1,600 lb. is noted. It remains to be seen whether the practicable weight-limit has yet been reached.

The mills mentioned in this paper and referred to specially in the tables are situated as follows:

1. Colorado mill.[1]
2. Combination, Goldfield, Nev.[2]
3. Desert, Millers, Nev.[3]
4. Dos Estrellas, El Oro, Mexico.[4]
5. El Oro, El Oro, Mexico.[5]
6. El Rayo, Santa Barbara, Chih., Mexico.[6]
7. Goldfield Consolidated, Goldfield, Nev.[7]
8. Guanajuato Consolidated, Guanajuato, Mexico.[8]
9. Guanajuato Development Co. (Pinguico Mill), Guanajuato, Mexico.[9]
10. Guanajuato Reduction & Mines Co., Guanajuato, Mexico.[10]
11. Homestake, Lead, S. D.[11]
12. Loreto (Cia. Real del Monte y Pachuca), Pachuca, Mexico.[12]
13. Montana-Tonapah, Tonopah, Nev.[13]
14. North Star, Grass Valley, Cal.[14]
15. Palmarejo, Chihuahua, Mexico.[15]
16. San Prospero, Guanajuato, Mexico.[16]
17. San Francisco, Pachuca, Mexico.[17]

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[1] A mill in Colorado, with the practice of which I am familiar, but the name of which I am not permitted to give.

[2] *Mining and Scientific Press*, vol. xciii., No. 15, pp. 451 to 454 (Oct. 13, 1906).

[3] *Ibid.*, vol. xcv., No. 16, pp. 494 to 500 (Oct. 19, 1907).

[4] Private communication.

[5] *Mining World*, vol. xxvii., No. 17, pp. 699 to 703 (Oct. 26, 1907).

[6] *Engineering and Mining Journal*, vol. lxxxvi., No. 2, pp. 78 to 80 (July 11, 1908).

[7] *Ibid.*, vol. lxxxvi., No. 10, pp. 467 to 474 (Sept. 5, 1908).

[8] *Ibid.*, vol. lxxxv., No. 14, pp. 710 to 717 (Apr. 4, 1908); *Mining and Scientific Press*, vol. xcviii., No. 19, p. 656 (May 8, 1909).

[9] *Engineering and Mining Journal*, vol. lxxxvi., No. 21, pp. 997 to 1001 (Nov. 21, 1908).

[10] *Ibid.*, vol. lxxxvi., No. 13, pp. 615 to 620 (Sept. 26, 1908).

[11] *Trans.*, xxxiv., 590 to 598 (1904); *Mining World*, vol. xxviii., No. 8, pp. 323 to 324 (Feb. 22, 1908); *Mines and Minerals*, vol. xxvii., No. 8, pp. 358 to 363 (Mar., 1907).

[12] *Engineering and Mining Journal*, vol. lxxxvi., No. 14, pp. 651 to 652 (Oct. 3, 1908); *Mexican Mining Journal*, vol. vii., No. 4, p. 16 (Oct., 1908).

[13] *Mining and Scientific Press*, vol. xcvii., No. 10, pp. 324 to 327 (Sept. 5, 1908).

[14] Private communication.

[15] *Trans.*, vol. xxxvi., 246 (1906).

[16] *Mining and Scientific Press*, vol. xcvii., No. 4, pp. 130 to 132 (July 25, 1908).

[17] *Engineering and Mining Journal*, vol. lxxxvi., No. 14, pp. 652 to 653 (Oct 3, 1908).

18. Standard, Bodie, Cal.[18]

19. Veta Colorado, Parral, Chih., Mexico.[19]

In Table I. are given the details of modern stamp-mill equipment.

TABLE I.—*Details of Stamps in Modern Mills.*

Name of Mill.	No. of Stamps.	Weight of Stamps.	Height of Drop.	Drops.	Duty Per 24 Hr.	Screen.	Life of Die.	Life of Shoe.	Life of Screen.
		Lb.	In.	Per Min	Tons	Mesh.	Days	Days	Days.
Colorado	60	1,050	6 to 8	100	3.8	26	50	112	3
Combination	20	1,200	6	108	4.5	....	74	96	10
Desert	100	1,050	6	104	4.79	12–14	59	76	30
Dos Estrellas No. 2	120	1,250	6.5	102	4.2	16 & 26	65	65	2 to 5
El Oro	{ 100	1,000	7.5	104	3.75	} 35			
	{ 100	1,150	6	102	4.00				
Goldfield Consol	100	1,050	....	108	......	16			
Guanajuato Consolidated	80	1,050	7.5	104	3.6	50	......	......	30–35
Guanajuato Devel. (Pinguico)	40	1,050	6.5	104	6.25	2, 4 & 8			
Guanajuato Reduction	160	1,050	7.5	100	3.1	26			
Homestake	1,000	900	10.5	88	4.0	No. 8 slot			
Loreto	40	1,050	....	106	3.0	16			
Montana-Tonopah	40	1,050	7	100	3.5	20			
North Star	80	1,050	8	96	3.1	20	......	......	25
San Francisco	30	1,050	6.5	104	......	20			
Standard	20	1,000	4 to 6	96–106	2.3	80	57	122	55
Veta Colorado	100	1,050	7	....	......	8–10			

3. *Mills.*—In a few cases Bryan and Huntington mills are used for crushing, though always as an intermediate grinder. The Combination mill uses Bryan mills to grind the coarse product from the stamps, taking a 12-mesh product and crushing through a No. 9 screen. A sizing-test on this product shows a large percentage passing through 100-mesh screen.

The Colorado and Desert mills follow the stamps with Huntington mills, while Dolores, Esperanza, Guanajuato Development, and Loreto, use Bryan or Chilean mills in a similar place. At El Rayo Huntington mills are used to grind the product from the fine rolls.

4. *Tube-Mills.*—Tube-mills are being used more and more for fine grinding in many of the mills herein described. In the majority of cases it has been found that regrinding all of the material to pass 100- or 150- or even 200-mesh screens permits

[18] Private communication.

[19] *Engineering and Mining Journal*, vol. lxxxvi., No. 3, pp. 120 to 122 (July 18, 1908).

a higher extraction with cyanide, especially on silver-ores, and when the increased cost of regrinding is more than balanced by increased extraction, such a procedure warrants careful investigation. Caetani and Burt[20] have devised a method of determining to what extent it is permissible to regrind in tube-mills, using "sand-indexes" as a means of comparison. Sizing-tests are made on all of the products which are to be compared, and the weight-percentages are multiplied by the respective extraction-percentages. These values are tabulated, giving the probable total extraction-value for the samples examined. The increased extraction of reground products can thus be determined and balanced against the increased cost of regrinding; in this manner the economic limit of regrinding can be determined. For El Oro ore, the limit was found to be a product sizing as follows: above 60-mesh, 0.10 per cent.; from 60- to 80-mesh, 2.9 per cent.; from 80- to 100-mesh, 8.6 per cent.; from 100- to 150-mesh, 52.5 per cent.; from 150- to 200-mesh, 12.5 per cent., and below 200-mesh, 24.2 per cent. The best grade of pebbles, known as Danish pebbles, come from Greenland and cost from $16 to $20 per long ton at New York. French pebbles cost from $9 to $12 per long ton, but do not wear as well. The consumption per ton of ore is from 2 to 8 lb., depending upon the character of the ore and the amount of regrinding. In several mills the pebbles have been replaced partly or in whole by pieces of hard quartz,[21] which is cheaper, especially if the quartz carries gold and silver.

The present lining is generally of cast-iron, wrought-iron, soft-steel, or silex. At El Oro the lining consists of plates of iron cast in the form of channels, which are bolted in the shell of the tube-mill with the channels running lengthwise of the shell. The pebbles become wedged in these channels and form a lining which takes most of the wear and lasts about 10 months.

At the Standard mill the lining is soft-steel, which lasts about 10 months, with a consumption of about 0.5 lb. of steel per ton of ore.

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[20] *Trans.*, xxxvii., 3 to 55 (1907).

[21] *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. vi., No. 10. p. 312 (Apr., 1906).

Brown[22] considers the wear of pebbles and liners to be a function of time rather than of tonnage. The proportion of solids in the feed also has a marked effect on the consumption.

Table II. gives details of construction and operation of modern tube-mills.

TABLE II.—*Details of Modern Tube-Mills.*

Name of Mill.	Make.	Number.	Size.		Rev. Per Min.	Capacity Per 24 Hours.	Power.	Pebbles.		Lining.	
			Diam.	Length				Kind.	Consumption Per Ton Ore.	Kind.	Consumption
			Ft. In.	Ft. In.		Tons.	H.P.		Lb.		Lb.
Combination	Abbé.	1	4	16	26	80	16	Danish.	2.4	Silex.	1.2
	Abbé.	1	4	12	26	24	14				
Dos Estrellas	A-C.	5	5	24	26	121	55	Quartz.	....	El Oro.	0.84
El Oro	Abbé.	2	4—6	19—6	31	100	50	....	8.4	El Oro.	1.0
	Krupp.	1	8—11	19—6	31	110	48				
	Krupp.	1	4—11	23—0	25	190	80				
	Krupp.	1	4—11	26—0	27	275	87				
Goldfield Consol	Gates.	6	5	22							
Guanajuato Reduct.	Abbé.	2	4—6	20	...	80	43	Danish.	0.75	Silex.	
Loreto	Abbé.	2									
	Krupp.	1									
Montana-Tonopah	Gates.	2	5	22	27	52	42.5	....	2.22	Silex.	
North Star	Abbé.	1	4—6	20	20	80	16	Quartz.	40	Chilled Iron.	
San Francisco	Krupp.	...	4—3	18—1	29	....	....	Danish.	....	El Oro.	
Standard	Gates.	1	5	22	24	125	50	Danish.	3.0	Soft-Steel.	0.5
Veta Colorado	....	5	5	14	...	....	....	....	....	Chilled Steel.	

II. CLASSIFICATION.

When concentration is followed by cyaniding it is not necessary to obtain a well-sized product, hence screens are supplanted almost entirely by classifiers. It is not so much desired to secure a sized product as to obtain a product of which one constituent will allow percolation; the other can be treated separately, usually by agitation and classification, which is the simplest and least expensive way. The separation into two products, one of which will settle rapidly, the other less so, is the desideratum, and since the extraction obtained by concentration is not high, nor need it to be so, sizing is entirely eliminated.

The classifiers in general use are the V- and conical *spitzkasten*, hydraulic cone-classifiers, and mechanical thickeners of the Dorr type. Owing to the excessive space and head-room

[22] *Mining and Scientific Press,* vol. xciii., No. 9. pp. 261 to 262 (Sept. 1, 1906).

required, as well as inefficiency, the V-*spitzkasten* are being rapidly replaced by the other types. Cone-classifiers are generally used to separate the table-product from that of the vanner, to separate sand going to the collecting-tanks from slime, and to separate sand going to the tube-mills from the slime which is already sufficiently fine.

Strictly speaking, the Dorr classifier[23] is a settling-machine in which the material settling out is removed by mechanical scrapers. These machines occupy much floor-space, but there is very little loss in head-room, and their results are more satisfactory than those of *spitzkasten* or cone-classifiers. The weak parts which appeared in the earlier designs are now replaced by stronger parts, so that the cost of repairs is not heavy, and very little attention is required to keep them running satisfactorily.

III. Amalgamation.

Amalgamation is used in many mills, and in a few cases a large percentage of the gold is recovered on the plates. At the Homestake plant from 70 to 75 per cent. saving is made on the plates and in the batteries.

With an ore containing little gold, amalgamation is seldom attempted. With a gold-ore in which a large part is free gold, or in which the gold is in coarse particles, plates are generally used. If the material is to be reground, there is no need for plates or concentrators.

At the Combination mill outside-amalgamation is used. On the outside, a splash-plate first receives the pulp, which falls to a lip-plate 12 in. wide, and then passes to the apron-plate, 54 by 120 inches.

IV. Concentration.

Tables are used in 11 mills of the plants reviewed in this paper, vanners in 8, and both tables and vanners in 6 mills. By concentrating a large part of the heavy minerals a product is obtained which can be shipped to the smelter and a high saving obtained, though at a high cost. In some ores it is found that concentration removes a large part of the cyanicides, which thus reduces the consumption of cyanide in the leaching- and agitation-tanks.

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[23] *Mines and Minerals*, vol. xxviii., No. 11, p. 541 (June, 1908).

When the ore is crushed in solution, the floors beneath the tables and vanners should be of cement, and have sufficient slope to drain to launders, in order to prevent loss of metals already in solution.

It is questioned whether it is advisable to concentrate when the precious-metal content of the ore can be removed by cyaniding. If the gold is coarse, especially if amalgamation is not used, concentration may be advisable.

In all of the Goldfield and Tonopah mills some form of concentration is used. The Desert, Goldfield Consolidated, and Montana-Tonopah mills employ tables, while the Combination and Montana-Tonopah use vanners also. Experiments are now under way which seem to indicate that these concentrates can be profitably treated on the ground instead of at the smelters. In Guanajuato and at Pachuca tables are widely used. Vanners are used at El Rayo, but tests are being made with the idea of dispensing with them entirely, since the handling of the concentrates is a source of much annoyance, and the cost of smelting is quite high. At the San Prospero mill, Guanajuato, 16 cement tables, 56 in. by 27 ft., with a slope of $\frac{1}{8}$ in. per foot, having scratched riffle-surfaces, are used. The ratio of concentration is 2,600 : 1, making an extraction of 3 per cent. of the gold and silver contained in the ore. The 8 Wilfley tables at the same place concentrate 230 : 1, and save 24 per cent. of the gold and silver.

At the El Oro mill a part of the coarse sand is passed over 20 ft. of blanket sluices, which remove from 0.25 to 0.5 ton per day. These concentrates contain 9 per cent. of metallic iron, which contains from $150 to $200 of gold and 50 oz. of silver per ton. These concentrates are mixed with 1 per cent. of lime and leached for 30 or 40 days in a 0.2-per cent. cyanide solution, extracting 98 per cent. of the gold- and 94 per cent. of the silver-content.

Table III. presents details of operation of tables and vanners.

V. Collecting—Settling—Decanting.

1. *Sand-Collecting.*—The general practice of sand-treatment begins with settling in collecting-tanks, draining off the surplus water or solution, and transferring the sand to the treatment-

TABLE III.—*Details of Tables and Vanners.*

Name of Mill.	Tables.	Vanners.	Concentration-Ratio.	Recovery.	
				Silver.	Gold.
				Per Ct.	Per Ct.
Colorado	11	30	13–18 to 1		
Combination	....	7	40 to 1	22.3	
Desert	25	9	90 to 1	15.63	30.32
El Rayo	....	16			
Goldfield Consol	70				
Guanajuato Consol	16	13	50 to 1	50	
Guanajuato Reduct	32	18	....	31.0	28.0
Loreto	8	30			
Montana-Tonopah	8	16			
North Star	9				
San Francisco	6				
Veta Colorado	24				
San Prospero	8	....	230 to 1	24	

tanks. In some cases a preliminary treatment with cyanide is given in the collecting-tanks, but generally cyaniding is done in the leaching-tanks.

The time required for treatment of sand, as compared with lime, is very long, due chiefly to insufficient aëration. Consequently, the preliminary treatment in the collecting-tanks is not to be condemned, since the transferal opens up and exposes the charge to the air.

The sand in the collecting-tanks is removed by means of a shovel or a Blaisdell excavator, the use of the latter device reducing the cost of handling very much. Cars and belt-conveyors are used to transfer the sand to the leaching-tanks, into which it is distributed by shovel or a Blaisdell distributer. Generally, a Butters and Mein distributer is used to distribute the charge in the collecting-tank.

The Combination mill uses a Gould pump to assist in the drainage of the surplus moisture which remains after the distributer is stopped. The Desert mill distributes the sands and solution into the tanks with a Butters and Mein distributer, which is suspended by an overhead trolley, and can be swung to any one of the four tanks. Roller blind-overflow gates are provided, over which the surplus solution is carried. It requires a period of 24 hr. to drain, after which a vacuum is applied to complete the draining. At the El Oro mill the collecting-tanks are also provided with a Butters and Mein dis-

tributer. The overflow is handled by vertical slats, against which canvas curtains are unrolled as the tank fills up. At the Palmarejo mill a large masonry tank of four compartments is used; each compartment is 25 by 80 by 4 ft. in dimensions, and the total capacity is 32,000 cu. ft.; 2-in. planks are used to raise the height of the overflow as the compartment fills up. Each compartment holds from 48 to 60 hr. of mill-product.

2. *Slime-Collecting.*—In a few plants, especially if filter-presses are not used, several settlings of the slime are required in order to remove the desired quantity of the precious metals that are already in solution. This settling occupies much time and requires extra tank-capacity; hence, any means of hastening the operation commands due consideration.

When crushing is done in water, it is necessary to remove a large part of the water in order to prevent too great a dilution of the cyanide solution and to avoid the accumulation of solution, which eventually must go to waste. With the exception of the practice at the Homestake mill, filter-pressing has not reached the point at which it would pay to filter this slime before treatment, so settling and decanting is generally used, which reduces the quantity of moisture to about 50 per cent. of the pulp. After the percentage of moisture has been sufficiently reduced, strong cyanide solution is added or the solution is brought up to strength by the addition of the dry salt. The pulp may then be treated direct in the collecting-tank, giving it the proper agitation and aëration, or it may be transferred to an agitation-tank for treatment. In this latter case, it is better to add the solution and the cyanide to the agitation-tank, since some of the solution containing gold and silver, which was left in the collecting-tank from a previous charge, will be lost during the subsequent filling and decantation. If the water thus removed be always returned to the batteries, this matter will not be a serious one, providing there are no leaks in the launders and sumps.

3. *Details of Treatment.*—At the Combination mill conical-bottom settlers with a rim overflow are used, the slime being left with about 50 per cent. of water and a specific gravity of 1.4. Sufficient salt is added to give a strength of 0.075 per cent. of KCN after it is transferred to the agitator.

At the Desert mill collecting-tanks with overflow-rims are

used. Part of the overflow goes to three sumps, 36 by 8 ft., whence it is returned to the battery storage-tank, and part is decanted for precipitation. The pulp is transferred from collecting-tanks by 8-in. centrifugal pumps without interrupting the operation of collecting.

At Goldfield Consolidated mill 16 settlers, 29.5 by 12 ft. in size, with overflow-rims and false conical bottoms, are used. The thickened pulp is pumped to the Pachuca tanks, the overflow going to two clarifying-tanks, 34 by 10 ft. in size, also provided with overflow-rims and false conical bottoms.

At the Guanajuato Consolidated mill the slime from the classifier is thickened in two parallel rows of masonry tanks, 12 compartments in a row, each 9 by 9 by 8 ft. in size, supplemented by three *spitzkasten*, each 8 by 8 by 30 ft. in size, divided into five compartments. The thickened pulp contains about 75 per cent. of moisture.

At the Pinguico mill of the Guanajuato Development Co., the slime is also collected in three tanks, 30 by 11.5 ft. in size, provided with overflow-rims. The thickened pulp is drawn off continuously from the bottom through three 3-in. pipes and flows to the treatment-tanks.

Merrill filter-presses are used at the Homestake mill, first for removing the moisture from the slime, and later for the cyanide treatment in the same presses. This simple method is to be recommended when the nature of the slime will permit.

The Montana-Tonopah uses three settling-tanks, 30 by 10 ft. in size, provided with a false bottom sloping 12°, and with leveling-rim, overflow-launder, and decanting-appliances. The pulp, when sufficiently thickened, is pumped to the agitators.

At the San Francisco mill the pulp goes to the Pachuca tanks, provided with overflow-launder, and is thickened to about 67 per cent. of moisture. After agitation, it is sometimes decanted down 4 or 5 ft. below, flowing to the filter-presses.

Agitation at the Standard Consolidated mill takes place during and immediately after filling the tanks, but before pumping to the filter-tanks it is decanted down about 3 feet.

Table IV. contains data of dimensions and capacity of collecting- and setting-tanks.

TABLE IV.—*Details of Collecting- and Settling-Tanks.*

Name of Mill.	No.	Size.	Total Capacity.	Capacity Per 24 Hr.
		Ft.	Cu. Ft.	Cu. Ft. Per Ton.
Sands.				
Colorado	3 4 1	40 by 6 23 by 6 30 by 8	38,247	165
Desert	4	33 by 7.5	25,659	80
Dos Estrellas	8	22 by 7	21,288	110
Guanajuato Reduct.	3	40 by 8	36,159	139
North Star	6	14 by 10	8,640	170
Palmarejo	1	100 by 80	32,000	
Slimes, settling.				
Combination	4 2	12 by 6 16 by 9	6,333	97
Desert	4	36 by 20	81,430	480
Dos Estrellas	cone	34 by 35	10,592	28
El Oro	4	36 by 20	81,430	215
Goldfield Consol.	16	29.5 by 12	131,230	230
Guanajuato Consol.	24 3	9 by 9 by 8 8 by 8 by 30	18,432	125
Guanajuato Develop.	5	30 by 11.5	40,644	225
Guanajuata Reduct.	3	18 by 22	16,797	70
Homestake	9 9	18 by 20 26 by 20	141,372	90
Montana-Tonopah	3	30 by 10	21,206	150

VI. LEACHING.

Usually the sand, conveyed from the collecting-tanks to the leaching-tanks by cars or belt-conveyors, is distributed by shovel or by mechanical distributers. Since the slime is largely removed from the sand there is little contained water, and it is not so important to secure as uniform a distribution if treated in separate leaching-tanks as when treated in the collecting-tanks.

The quantity of lime necessary to secure the desired alkalinity, varying from 0.1 to 5 lb., is added during the transfer, also the lead acetate, when used.

The time of treatment varies from 5 to 16 days, depending upon the character of the ore and the fineness of grinding. Seldom are these limits exceeded.

The first treatment is usually with strong cyanide solution, containing from 0.2 to 0.45 per cent. KCN. The solution is added in separate charges, so as to allow the air to penetrate through the sands after each percolation, and is followed by weak-solution washes, then water, and finally the residue is sluiced, trammed by car, or carried by belt-conveyor to the dump. In some cases the order of the strong and weak solu-

tions is reversed, in others strong solution alternates with weak. The strength of weak solution varies from 0.015 to 0.3 per cent. KCN. Beneficial results have been claimed by adding the solution at the bottom as well as at the top, but it should be done slowly in order to avoid channeling the charge. In one instance, the addition of solution from the bottom increased the extraction 20 per cent. The greatest drawback to this method is the difficulty in securing sufficient aëration, and I am in doubt whether the numerous changes of solution are as beneficial as the better aëration resulting from these changes. In some cases the lower layer of sand contains more silver than the upper portions, which can be attributed largely or entirely to the lower degree of aëration.

The percentage of extraction from the sand is usually lower than that from the slime of the same ore, which is permissible, if the difference in extraction is less than the cost of regrinding and handling the extra amount of slime.

At the Desert mill the sand is transferred to the leaching-vats by a Blaisdell excavator, and during this time 0.10 lb. of lead acetate is added per ton of sand. The lime is fed to the ore as it passes through the gyratory crushers, averaging 14.92 lb. per ton of ore; 30 tons of 0.5-per cent. NaCN solution is added to the top of charge, which saturates it; the valves being closed for 12 hr., the strong solution is circulated through each charge for 16 hr. by means of small air-lifts attached to each tank; drained, and repeated pumpings of solution of from 0.2 to 0.25 per cent. of NaCN are applied for 100 hr.; drained 24 hr.; transferred to another leaching-vat with addition of 0.0716 lb. of lead acetate per ton of sand; one application of 0.35-per cent. NaCN solution, followed by repeated pumpings for 96 hr. of weak solution, followed by water-wash; drained, and discharged by excavator. Total time, including collecting, transferring, and discharging, from 288 to 360 hr. All solutions are drained below the top of the charge before another is added. The tailings assay 0.03 oz. of gold and 3.10 oz. of silver per ton.

At the Dos Estrellas mill the sands are first leached with 0.22-per cent. solution, followed by 0.47-per cent. solution, then by 0.12-per cent. solution, and are discharged by Blaisdell excavator and belt-conveyor.

At the El Oro mill, after filling the collecting-tank and drain-

ing off the surplus, three or four washes of 0.2-per cent. solution are given. The sand is then transferred to the leaching-tanks and given several washes with 0.2-per cent. solution, 8 hr. apart. The material is then transferred to a second leaching-tank and the treatment continued for 240 hr., after which the surplus moisture is drained off by a vacuum-pump, and the tailings discharged to the dump.

The quantity of sand treated daily varies from 160 to 200 tons, which contributes about 35 per cent. of the total quantity of the ore. One-half of the metals is extracted by the time the material reaches the collecting-tanks, and the extraction is 70.4 per cent. of the gold and 34.3 per cent. of the silver on reaching the leaching-tanks.

Later practice at El Oro is to slime the ore completely.

At the El Rayo mill filling a tank requires 36 hr., leaching 150 hr., and draining and emptying 24 hr., making a total of 204 hr. The quantity of ore treated as sand amounts to 65 per cent.

At the Guanajuato Consolidated mill the time required for the various operations is:

Filling the collecting-tanks requires	15 hr.
Draining,	15 hr.
Addition 20 tons of 0.3-per cent. solution and leaching,	12 hr.
Draining,	6 hr.
Addition 20 tons of 0.3-per cent. solution and 20 g. lead acetate per ton,	12 hr.
Draining,	12 hr.
Transferring to second-treatment tanks, 450 kg. of CaO added,	6 hr.
Addition 30 tons 0.3-per cent. solution and 25 kg. of KCN and leaching,	24 hr.
Draining,	6 hr.
Addition 30 tons of solution and leaching,	156 hr.
Addition weak (0.15-per cent.) solution, leaching and draining.	48 hr.
Addition of water and draining,	18 hr.
Sampling and sluicing,	6 hr.
Total treatment-time,	336 hr.

At the Homestake mill the tanks are first filled with water, then the sands are added and distributed by a Butters and Mein distributer with from 3 to 5 lb. of lime per ton. This filling takes 11 hr., and is followed by the addition of 0.14-per cent.

solution, with frequent drainage, requiring about 72 hr. Weak solution is then added during a period of 24 hr. Then follows a water-wash, which continues until the effluent solution contains only from 0.02 to 0.03 per cent. of KCN. Finally, the tank is sluiced out by two men using a 3-in. hose. The time required for the total treatment is about 150 hours.

Table V. gives details of leaching-tanks:

TABLE V.—*Details of Leaching-Tanks.*

Name of Mill.	Number of Tanks.	Size of Tanks.	Total Capacity.	Capacity Per 24 Hr.	Time of Treatment.
		Ft.	Cu. Ft.	Tons.	Hours.
Colorado	3 4 1	40 by 6 23 by 6 30 by 8	38,247	165	
Combination	16	16 by 5	16,085	180	288
Desert	18	33 by 8	123,161	335	305
Dos Estrellas	12	36 by 5.5	67,176	385	184
El Oro	12	40 by 6.5	98,018	165	360
El Rayo	....	25 by 6	....	....	210
Guanajuato Consol.	7[a] 15	26 by 5 26 by 6	66,367	490	336
Guanajuato Develop.	14	29.5 by 5	47,845	190	
Guanajuato Reduct.	15	40 by 8	150,795	580	480
Homestake	14	44 by 9	191,587	80	120
North Star	6	23 by 8	17,500	290	168

[a] Used for collecting-tanks also.

VII. AGITATION.

To allow the solution to come into sufficient contact with the metals and to provide sufficient aëration, it has been found necessary to agitate the pulp during the time of treatment. Mechanical stirring is largely used, assisted in some cases by circulation with centrifugal pumps, and also by introducing air into the pulp. Centrifugal pumps, used both alone and with air, secure good results. The most successful method is the use of a high conical-bottom tank, provided with a central air-lift, invented by Brown and first used in New Zealand. In America it is better known as the Pachuca tank.[24] This method of agitation is used in several of the most modern mills, and may, in time, supersede nearly all others.

Objection has been raised to the use of the Pachuca tank, because of a supposed increase in the consumption of cyanide,

[24] *Engineering and Mining Journal*, vol. lxxxvi., No. 14, p. 653 (Oct. 3, 1908).

due to the greater decomposition by the air, but tests have recently shown that the reverse is the case. With plenty of agitation and aëration it is possible to dissolve the metals to an economic degree in a much shorter time than with ordinary practice, which shortens the time that the cyanide is subjected to the decomposing influence of the air. The power required for these tanks is small, varying from 1 to 5 h.p. for a 15- by 45-ft. tank.[25] An initial presure of about 50 lb. is used, which is lowered to 25 lb. as soon as the pulp is in good circulation.

The Hendryx agitation-tank has also met with good success, securing good agitation and aëration with a moderate expenditure of power.

Before agitation is commenced, the necessary quantity of lime, and in some mills, lead acetate also, is added. From 12 to 30 hr. of agitation is given for gold-ores, while a much longer time, varying from 24 to 90 hr., is required for silver-ores.

At the Combination mill, after decanting to 50 per cent. of moisture, sufficient KCN is added to raise the solution to 0.075 per cent. The material is then transferred by a 3-in Krogh pump to the agitator, which has a steep conical bottom. Here it is agitated by pumping, which withdraws the pulp from the bottom and adds it at the top. After agitation for from 12 to 18 hr., it is transferred to the slime-reservoir.

At the Desert mill the agitating-tanks, numbering seven, are provided with mechanical agitators driven by a 40-h.p. motor, with gearing and friction-clutch over each tank. These agitators make 5 rev. per min. Additional agitation is secured by means of four 12-in. square air-lifts, made of 1- by 12-in. boards, reaching to within 6 in. of the bottom of the tank. Air is introduced at the bottom of the lift through a 0.75-in. pipe. The air-lifts are in the four quadrants of the tank, and each directs its flow inwardly by means of launders extending towards the center of the tank. About 100 tons of barren solution, to which sufficient cyanide has been added to bring it to a strength of 0.45 per cent. NaCN, is first introduced before the pulp is transferred to it from collecting-tanks. The specific gravity of the final mixture is about 1.144, or 21 parts of slime

[25] *Engineering and Mining Journal*, vol. lxxxvi., No. 14. p. 653 (Oct. 3, 1908); No. 19, p. 901 (Nov. 7, 1908); *Mexican Mining Journal*, vol. vii., No. 4, p. 18 (Oct., 1908).

to 79 parts of solution. At the beginning of agitation 40 lb. of lead acetate is added to the charge. At the end of 40 hr., 20 lb. of lead acetate is added. The total time of agitation is 56 hr., at the expiration of which it is discontinued and the contents of the tank are allowed to settle, and an average of 180 tons of clear solution is decanted for precipitation, after which the thickened pulp is transferred to the Butters filter stock-tank for final treatment and a new charge is made up. The alkalinity of the solution in the treatment-tank is maintained at 1 lb. per ton of protective alkali in terms of calcium oxide.

At the El Oro mill the quantity of slime amounts to from 550 to 650 tons per day, of which a sizing-test shows to be 20.5 per cent. 150-mesh, 4.5 per cent. 200-mesh, and 75 per cent. through 200-mesh. On reaching the collecting-tanks 80 per cent. of the gold and 13 per cent. of the silver have been dissolved. Mechanical and pump-agitation is given for 6 hr., during which time 22 lb. of NaCN and 22 lb. of lead acetate are added. Near the end of the agitation, from 130 to 170 lb. of slaked lime is added, then settling and decanting for 8 hr., then five washings given; finally the pulp passes to the settling-tanks.

At the El Rayo mill the pulp is agitated for 24 hr. by mechanical stirrers, assisted by air.

At the Goldfield Consolidated mill the Pachuca tanks receive the pulp, thickened to the proper consistency in the settlers, and after thorough agitation the contents are pumped to the slime-reservoirs above the Butters presses.

At the Guanajuato Consolidated mill the slime is transferred from the collecting-tanks to the agitation-tanks, with the addition of 125 lb. of lime and 17.5 lb. of lead acetate to each charge. The solution is brought up to a strength of 0.1 per cent. KCN by the addition of the dry salt. The pulp, consisting of 1 part of dry slime to 5 parts of solution, is agitated for 24 hr., after which it is settled and 36 in. of the clear solution decanted. An addition of 36 in. of solution is made and the pulp agitated, then again settled and decanted. In a similar manner six additional agitations and decantations are given, followed by two washes of water. The pulp then passes to the high settlement-tanks, where it is settled and decanted to a water-content of 70 per cent. After the Burt filter installation is completed, the routine of slime-treatment will be changed.

At the Guanajuato Development mill (Pinguico), the first treatment, consisting of 6 hr. of agitation, is given in the collecting-tanks. The pulp is then transferred to a 30- by 11.5-ft. agitation-tank, and agitated for 24 hr. by means of air-jets in the bottom of the tank, assisted by a 6-in. centrifugal pump. It is then allowed to settle 8 hr. and decanted. Three 4-hr. agitations in 0.2-per cent. solution follow, with intermediate settling and decantation; finally the pulp is transferred to the slime-reservoir. The total time of treatment is about 100 hr.

At the Homestake mill the only agitation given is that received while the pulp is being transferred to the filter-presses.

At the Loreto mill, after filling the agitators, the excess solution is decanted and the tank filled with 0.15-per cent. KCN solution, and sufficient KCN added to raise the strength to 0.20 per cent.; then 0.7 lb. of lead acetate per ton of slime is added, and agitation for 36 hr. is given; the pulp is then settled and the clear solution decanted. A second agitation of 6 hr. is given, followed by decantation. Then a third agitation of 6 hr., the pulp being finally pumped to the filter-plant.

At the Montana-Tonopah mill the pulp is transferred from the settling-tanks by a 6-in. centrifugal pump to two of six Hendryx agitators, agitated in 0.2-per cent. NaCN solution for 32 hr.; finally transferred to the slime-storage tank. The power required to agitate a charge of 100 tons of pulp amounts to 6 horse-power.

At the San Francisco mill the pulp is agitated in five 15- by 45-ft. Pachuca tanks in 0.45-per cent. solution, to which 0.7 lb. of lead acetate per ton has been added, and then transferred to the filter-tanks; 1.1 h.p. is required for each tank of 50 tons.

At the Standard Consolidated mill, commencing with the filling of each tank, the mechanical stirrers are started and continued for 12 hr. after the tank has been filled. Compressed air, also used to assist the agitation, is conducted to the bottom of the tank through pipes along the side of the tank. After settling and decanting, the pulp is transferred to the filter-tanks.

At the Veta Colorado mill seven Pachuca tanks now being erected will be used for agitation.

Details of agitation-tanks are given in Table VI.

TABLE VI.—*Details of Agitation-Tanks.*

Name of Mill.	No. of Tanks.	Size.	Total Capacity.	Capacity Per Ton Slime.	Time of Agitation.
		Ft.	Cu. Ft.	Cu. Ft.	Hours.
Combination	1 2	12 by 18 10 by 14	4,234	65	16
Desert	7	36 by 20	142,500	838	30
Dos Estrellas	12 6	36 by 10 36 by 20	243,296	750	13.6
El Oro	15	34 by 12	163,440	270	56
El Rayo	......	25 by 16	......	......	24
Goldfield Consol	10	15 by 45	62,500	110	
Guanajuato Consol	14	30 by 10	98,960	660	300
Guanajuato Reduct	13	36 by 12	158,789	660	72
Loreto	27	30 by 10	190,852	1,270	54
Montana-Tonopah	6	17 by —	......	......	32
North Star	2	8 by 7	1,500	30	
San Francisco	5	15 by 45	31,250	......	24
Standard	5 5	20 by 11 18 by 15	36,264	290	18
Veta Colorado	14	15 by 45	87,500	175 (est.)	

VIII. FILTRATION.

During the past five years a great advance has been made in the filtration of very fine material, and with the introduction of the vacuum-filter a new field has been opened for the cyaniding of gold-ores, and especially silver-ores. Formerly the old pressure-filters gave a small capacity and generally high costs, and the future of slime-treatment gave little promise, especially with low-grade ore. To-day, on account of the ease of modern filtering-methods, due consideration is given to the so-called "all-sliming" processes before installing a plant of any considerable capacity.

Butters and Moore filters are those generally used. The Ridgeway is being installed at the Veta Colorado mill. The Oliver, a continuous vacuum-filter, is in use at the North Star mill, and the Burt pressure-filter at the El Oro and Dos Estrellas mills.' The Merrill press is handling the Homestake slimes with the best of results. There is also a Merrill press at the El Rayo mill, which, however, is not in use at present.

For handling the slimes from the zinc-dust precipitation-plant the Merrill press with triangular form of plates is generally used. At the Combination mill the plant consists of two slime-storage tanks, two wash-water tanks, a 58-frame Butters filter, one acid-box for washing the frames, one vacuum-pump, two 4-in. Butters centrifugal pumps, one 3-in. Krogh centrifugal

pump. The quantity handled is 65 tons per 24 hr., making a 0.75-in. cake with a 22-in. vacuum. The washing is done with about 10 tons of 0.03-per cent. solution. The washed cakes are discharged by forcing in water from the inside of the frame. In cleaning the leaves of lime present it was found better to draw a 2-per cent. HCl solution through the leaves for 15 or 20 minutes.

At the Desert mill the filter is composed of 172 leaves in constant service. During the first 5 min. the solution drawn through is turbid and is returned to the circulation-pump. The discharged slime-cake is broken up by 1-in. jets of water under 70-ft. head directed into the hopper of the filter-tank. Air and water, formerly used for discharging, broke the stitching in the leaves. The filters are now discharged in the wash-solution by admitting water under 12-ft. head. The cakes are allowed to settle for 5 min., and the water run back to within about 6 in. of the thick slime in the hopper. The discharged slime contains 60 per cent. of water. The quantity treated averages 165 tons per 24 hours.

At the Guanajuato Development mill the Butters filter contains 80 leaves. A cake 1 in. thick, containing 40 per cent. of water, formed in about 3 hr. with 20-in. vacuum, is discharged by compressed air at 5 lb. pressure. About 35 h.p. is required to operate the plant, handling about 70 tons of dry slime daily.

At the Homestake mill are 24 Merrill filter-presses, each containing 92 plates, 4 by 6 ft. The cycle requires 10 hr. One man tends to 8 presses and one fills and sluices out the 24 presses, making four men per shift. The extraction is about 90 per cent., at a total treatment-cost of $0.25 per ton.

At the Montana-Tonopah mill 144 Butters leaves are in use. A charge of pulp is delivered to the filter-box, and after removing the solution it is given the necessary washes, the pulp being finally discharged through the bottom of the hopper and sluiced to waste. A cake from 1.25 to 1.5 in. thick is made in 3 hr., with a total output per day of 140 tons of dry slime. The solution is collected in a small storage-tank and pumped through the clarifying-press into any of the three precipitation-tanks.

At the Standard Consolidated mill a Moore vacuum-filter is in use containing two baskets, each having 45 leaves, 5 by

16 ft. The two filter-tanks and the wash-solution tanks are 24 by 7 ft. in size. The filter-tanks were formerly percolation-vats. During filtration a vacuum of 13 in. is maintained, though at first the relief-valves on the pumps must be opened so as to obtain a vacuum of only 7 or 8 in., since the flow of the solution is too great for the motor to handle. The pulp in the filter-vats is kept in agitation by sweeps operated by a shaft passing through the bottom of the tank. Air also is used, drawing the pulp from the bottom of the tank and discharging it at the top. After washing in 0.07-per cent. KCN solution the cake is discharged by compressed air. The cakes average 0.83 in. in thickness and, when discharged, contain 27 per cent. of water. The loss of cyanide due to this contained water is 0.38 lb. per ton of ore; 110 tons of slime are handled daily by the two basket-filters.

Details of size and capacity of filter-presses are given in Table VII.

TABLE VII.—*Details of Filter-Presses.*

Name of Mill.	Make of Press.	No.	Size of Leaf.	Total Area.	Area Per Ton.	Time of Cycle.
			Ft.	Sq. Ft.	Sq. Ft.	Hours.
Combination..........	Butters.	54	5 by 10	4,708	72	227
Desert..................	Butters.	172	4.3 by 9.75	14,448	84	180
Dos Estrellas..........	Burt.	70	28.3 sq. ft.	1,984	....	37
El Rayo................	Butters.	60	5 by 10	5,100	146	240
Goldfield Consol......	Butters.	336	5 by 10	28,560	50 (est.)	
Guanajuato Devel...	Butters.	80	5 by 10	6,800	97	180
Guanajuato Reduct.	Butters.	75	5 by 10	6,375	40	90
Homestake............	Merrill.	2208	4 by 6	52,992	32	600
Liberty Bell...........	Moore.	268	6 by 8	21,869	62	135
Loreto..................	Butters.	104	5 by 10	8,840	74	120
Montana-Tonopah..	Butters.	144	5 by 10	12,240	85	
North Star............	Oliver.	....	10 by 7 ft. revolving drum.	220	4.4	5.5
San Francisco.........	Butters.	....	....	....	....	180
Standard...............	Moore.	90	5 by 16	12,240	111	360
Veta Colorado........	Ridgeway.	130	4 sq. ft.	520	1 (est.)	

IX. PRECIPITATION.

Precipitation in America is done with zinc, either shavings or dust. The majority of plants use shavings, but the convenience and economy of the dust will probably cause most of the new plants to use it, and possibly many of the plants now using shavings will in time change over to dust.

The consumption varies from 0.6 to 1.5 lb. of zinc per ton of ore, with an average of about 1.25 pounds.

At the Desert mill the solutions are pumped to 14 zinc-boxes by two 3-in. Byron Jackson centrifugal pumps, directly connected to 2-h.p. motors. Six boxes receive solution by gravity head. Fifteen cubic feet of zinc-shavings are added to each compartment, making a zinc-capacity of 105 cu. ft. for each box, or a total of 2,100 cu. ft. for all boxes; 2,600 tons of solution are precipitated daily, the barren solution assaying from a trace to 11 cents in value per ton.

At the El Oro mill all of the weak solution is precipitated in 20 large zinc-boxes, each having five compartments, four of which are filled with zinc. These compartments are 4 by 3 by 2 ft. deep. The strong solution from the sand is handled by three large zinc-boxes, and the medium solution by three smaller boxes having six compartments, each 2.5 by 2.5 by 2 ft. deep. The amount of weak solution precipitated per 24 hr. is 3,500 tons, and about 200 tons each of the strong and the medium solutions.

At the El Rayo mill zinc-dust is added from time to time to a small cone, into which flows a stream of cyanide solution. The overflow from the cone passes to the suction of the pump, which draws the solution from the gold-tanks. The solution and dust, thoroughly mixed, are pumped to a Merrill press, which is cleaned up once a month. The zinc-consumption is 1.2 lb. per ton of ore treated.

At the Goldfield Consolidated mill zinc-dust is added to two 25-inch cones, which overflow to the suction of two 7- by 9-in. Aldrich triplex pumps, where it becomes mixed with pregnant solution and is pumped to four Merrill precipitation-presses, each containing 30 triangular frames, 48 in. in size. From the presses the precipitate goes to the refinery.

At the Guanajuato Consolidated mill the weak and the strong solutions are precipitated in two sets of zinc-boxes. Samples of the solution are obtained from 0.25-in. pet-cocks on the pipes entering the zinc-room, and samples of solution leaving the zinc-boxes are similarly taken. Each compartment holds about 20 cu. ft. of zinc-shavings, or 8 lb. per cu. ft. of capacity, the total 56 compartments holding 1,120 cu. ft., or 8,960 lb. About 1.3 lb. of zinc is consumed per ton of pulp-treatment, or 0.08 lb.

per ton of solution; 4,030 tons of solution per day, or 15.5 tons per ton of ore, are precipitated.

At the Guanajuato Development mill the precipitate-room contains four 6-compartment strong-solution boxes and eight 6-compartment weak-solution boxes. About 500 tons of strong and 1,000 tons of weak solution are precipitated daily. A clean-up is made twice a week. The precipitate, worked through 100-mesh screens, passes to the clean-up sump. The short zinc is returned to the first two compartments of the strong-solution boxes.

At the Homestake mill the weak solution only is precipitated in the sand- and slime-plants. This solution assays $3 in value and contains 0.06 per cent. of KCN. It is run into tanks, where approximately 0.2 lb. of zinc per ton of solution, in the form of an emulsion, is added to the pump-suction (Merrill method), and then is pumped through two filter-presses, the effluent solutions assaying from 1 to 3 cents per ton in value, thereby securing a precipitation of 99 per cent. Forcing the solution through the adhering zinc-cake insures good precipitation. At the end of the month, when the clean-up is made, the gauge-pressure stands about 10 lb. Two men make a clean-up in 10 hours.

At the Montana-Tonopah mill the solution from the Butters filter-leaves is pumped through a 30-in. frame filtering-press, and passes to the precipitation-tanks, of which there are three, each 14 by 14 ft. in size. Zinc-dust, fed from a small feed-cone, passes to the suction of a 5- by 6-in. Aldrich triplex pump, which raises the solution to two Merrill presses, each having 30 triangular frames, 48 in. in size. About 550 tons of solution per day are handled by the presses.

At the San Francisco mill, in the zinc-room, are 10 boxes, each having five compartments, 3 by 3 by 2.5 ft., with bottoms sloping to one side. On cleaning up, the precipitate is rubbed through 60-mesh screens, and the short zinc returned to the boxes.

At the Standard Consolidated mill the pregnant solutions from the gold-solution tanks flow to the zinc-precipitation boxes, of which there are three rows for the weak solution, each row having 20 compartments, 14 by 14 by 16 in. in size, and for the strong solution six rows having nine compartments each,

14.5 by 15 by 16.5 in. in size, and four rows of ten compartments each, 16 by 17 by 23 in. in size. The strong solution before precipitation averages 0.11 oz. of gold and 0.3 oz. of silver per ton, and after precipitation, 0.003 oz. of gold and a trace of silver per ton. The weak solution before precipitation assays 0.06 oz. of gold and 0.27 oz. of silver, and after precipitation 0.0056 oz. of gold and a trace of silver per ton. A clean-up is made twice a month.

X. Refining.

At many cyanide plants the precipitates are shipped to the smelter and not reduced to bullion.

In the clean-up, which varies from twice a week to once a month, the precipitate is transferred from the zinc-boxes to the clean-up tank, in which it is settled and decanted. In many places the precipitate is passed through from 30- to 100-mesh screens, and the zinc shorts returned to the zinc-boxes. This practice reduces the quantity of acid required for dissolving out the unconsumed zinc, and sometimes no acid whatever is needed.

Sulphuric acid is added to the clean-up sump to dissolve the zinc. The liquid is then decanted and the slime washed and stirred, settled, and the supernatant liquid drawn off. After washing, the precipitate is usually filter-pressed as dry as possible, sometimes by forcing hot air through it. The dried precipitate is first roasted, if necessary, then mixed with the proper fluxes, and melted down in a furnace. The usual flux comprises from 20 to 30 parts of borax and 8 to 20 parts of soda per 100 parts of precipitate. At some plants the mixed flux and precipitate are briquetted before adding to the crucible —a practice which largely eliminates loss by dust.

Another method, called the Tavener process,[26] adds litharge and a reducing-agent, and collects the gold and silver in a lead button, which is later cupelled. This method also largely eliminates loss by dust.

At the Combination mill the precipitate is cleaned with sulphuric acid, washed, dried, mixed with the fluxes and melted in a Faber du Faur furnace fired with coke. At the Desert mill five

[26] *Engineering and Mining Journal*, vol. lxxxiii., No. 13, p. 608 (Mar. 30, 1907).

clean-ups are made each month. Six men, working 8-hr. shifts, clean the 20 boxes in three days. The zinc in the first two compartments is washed, and all precipitate passed through a 16-mesh screen for removing short zinc. The zinc in the lower compartments is rinsed with water. Washing takes place in the head compartment, and as much of the precipitate as can be made to flow is pumped by a 5- by 6-in. Knowles triplex pump through two Johnson filter-presses having 24- by 24-in. frames. The thick precipitate is bailed out into tubs, mixed with the precipitate from the presses and fluxed with 10 parts of soda and 20 parts of borax to 100 parts of dried precipitate. This is done in drying-pans placed in the muffles of the drying-furnace. By this means it is not necessary to pulverize the precipitate, and the loss from dust is thereby minimized. After drying, the fluxed precipitate is broken up to a proper size and charged into six Faber du Faur tilting-furnaces, equipped with graphite retorts, for melting into bullion. Coke is used for fuel, and the first pour is made 4 hr. after starting the fire. During the last three months an average of 5,420 lb. of dry precipitate, yielding 4,531 lb. of bullion, or 83.6 per cent. bullion in precipitate, has been melted in 25 hr., using the six furnaces. This result is the average of regular clean-ups and melts five times a month.

This mill is installing a cylindrical zinc-washer, rotated by a small motor, and a centrifugal pump, to pass the solution from the zinc-boxes during clean-ups. The suction of the triplex pump is to be connected with the washer, and a spray of water introduced through a hollow shaft into the cylinder containing the zinc. The triplex pump will pump the precipitate through the filter-presses, and the centrifugal pump will deliver the solution and detached precipitate in the compartments of the boxes to a tank, from which it will be pumped through the presses by the triplex pump when it is not being used on the washer. By this improvement it is expected to make a clean-up with four men in two days.

At the El Oro mill a clean-up is made four times a month. The precipitate, passing 60-mesh screens, is pumped into a filter-press, and the resultant cake is partly dried in the press with steam for 1 hr. It is still further dried by steam in a steam drying-receptacle, mixed with about 18 per cent. of borax-glass and 6 per cent. of calcined soda, briquetted under 2,000 lb.

pressure per square inch and melted. Coke is used in three furnaces and crude oil in three furnaces.

At the El Rayo mill the dried precipitate is shipped to the smelter.

At the Goldfield Consolidated mill the precipitate from the presses is dried in pans, partly roasted, ground with the fluxes in a 24- by 48-in. revolving barrel, and melted in four Faber du Faur furnaces equipped with oil-burners.

At the Guanajuato Consolidated mill the zinc-boxes are cleaned up three times a month, and the precipitate collected in the sump-tank, from which it is pumped to a Johnson filter-press. The cake from the press is mixed with the fluxes, consisting of 30 parts of borax and 18 parts of soda to 100 parts of precipitate, dried in large trays and melted in No. 300 graphite crucibles.

At the Guanajuato Development mill the precipitate is pumped from the sump-tank to a Shriver filter-press having twenty 24-in. frames. The cakes are dried in pans, mixed with the fluxes, and melted in graphite crucibles. The resulting bars are remelted, cast into bullion bars and shipped.

At the Homestake mill the precipitate, removed from the filter-press, is first treated with hydrochloric acid, followed by sulphuric acid. The precipitate is then washed, dried in a large steam drier and mixed with the fluxes. After briquetting it is melted down and the resulting lead button cupelled.

At the Loreto mill the precipitate is dried, fluxed, melted in No. 300 graphite crucibles, and cast into bullion bars 950 fine.

At the San Francisco mill the precipitate, after passing through 60-mesh screens, goes to a filter-press, in which the excess moisture is expelled. It is partly dried by air in the press, followed by drying in pans, then fluxed and melted. The resulting bullion is 950 fine.

At the Standard Consolidated mill the precipitate is hauled from the zinc-room in tubs to the acid-room, where the unconsumed zinc is dissolved with sulphuric acid. Nine washings and decantations are given to remove the acid, after which the precipitate is filtered on a blanket ejector, using steam to obtain a vacuum. It is then dried, fluxed, and melted, and then cast into bullion bars.

Table VIII. gives details of consumption of materials per ton of ore ground and cyanided.

TABLE VIII.—*Consumption of Materials Per Ton of Ore Treated.*

Name of Mill.	Die.	Shoe.	Tube-Mill.		KCN.	Zn.	CaO.	Pb $(C_2H_3O_2)_2$.
			Pebbles.	Lining.				
	Lb.	Lb.	Lb.	Lb.	Lb.	Lb.	Lb.	Lb.
Colorado	......	......	......	......	0.93	0.53	2.1	0.005
Combination	......	......	2.4	1.2	2.10	0.98	8.0	0.5
Desert	0.53	4.8	......	......	3.41	1.72	14.92	0.62
Dos Estrellas	......	......	......	......	1.17[a]	1.07	9.78	0.13
El Oro (sand)	......	......	8.4	1.0	2.8	1.2	12.0	
El Oro (slime)	......	......	......	......	0.75			
El Rayo	......	......	......	......	5.0	1.2	16.0	
Guanajuato Consol.	......	......	......	......	5.15	1.3	6.0	
Guanajuato Reduct.	......	......	......	......	1.70[a]			
Homestake (slime)	......	......	......	......	0.31	0.13	4.5	
Loreto	......	......	......	......	2.2[a]	......	8.8	
Montana-Tonopah	......	......	2.22	......	2.8	0.97	9.2	0.28
North Star	......	......	40.0	......	0.5	0.33	4.0	
Standard	1.16	0.63	3.0	0.5	1.16[a]	0.6	7.5	

[a] NaCN.

XI. Extraction and Costs.

I here present the following interesting data pertaining to the percentage of extraction of the precious metals and the cost per ton of ore treated at vanner mills. In computing the costs at the Dos Estrellas, Guanajuato Consolidated, and Guanajuato Reduction mills, the *peso* is valued at $0.50, U. S. currency.

Colorado Mill.—Gold extraction, 94 per cent.

Combination Mill.—Plate-extraction, 42.5 per cent. ; concentrate-extraction, 22.3 per cent. ; leaching-extraction, 14 per cent. ; slime-extraction, 16 per cent. ; total extraction, 94.8 per cent.

Costs :

Crushing,	$0.23
Stamping,	0.82
Concentrating,	0.24
Regrinding,	0.40
Leaching,	0.69
Agitation,	0.75
Filtering,	0.43
Precipitating,	0.28
Refining,	0.15
Total,	$4.04

Desert Mill.—Costs:

Crushing and conveying,	$0.096
Stamping,	0.410
Huntington mills,	0.244
Concentrating,	0.201
Elevating and separating,	0.131
Conveying and discharging sand.,	0.186
Leaching sand,	0.673
Agitating slime,	0.582
Filtering and discharging slime,	0.114
Precipitation,	0.225
Assaying,	0.063
Refining,	0.126
Storehouse,	0.047
Stable,	0.006
Foreman,	0.050
Surface-plant,	0.040
General expense,	0.020
Steam heating,	0.153
Total,	$3.367

Dos Estrellas.—Costs:

Crushing,	$0.45
Classifying,	0.005
Regrinding,	0.19
Leaching,	0.265
Agitating,	0.20
Precipitating,	0.145
Refining,	0.045
General expense,	0.05
Total,	$1.350

Guanajuato Consolidated—Costs, crushing and concentrating:

Labor,	$0.13	
Supplies,	0.10	
Power,	0.26	
Assaying,	0.015	
General expense,	0.015	
Hacienda expense,	0.025	
Insurance,	0.005	
		$0.55
Cyaniding:		
Labor,	0.205	
Supplies,	1.03	
Power,	0.13	
Assaying,	0.06	
General expense,	0.07	
Hacienda expense,	0.12	
		1.615
Total,		$2.165

The total milling- and cyaniding-costs of the Guanajuato Consolidated for the full year of 1908 are given as $2.30 per ton.

Homestake (Mar., 1908).—Cost of cyaniding slimes:

Labor,	$0.0753
Power and lights,	0.0197
Chemicals,	0.1152
Other,	0.0351
Total,	$0.2453

Guanajuato Reduction.—Gold-extraction, 88 per cent.; silver-extraction, 86 per cent. Cost: $1.575.

Montana-Tonopah.—Gold-extraction, 94 per cent.; silver-extraction, 87 per cent. Cost: precipitation, 0.119; cleaning-up, 0.0073; regrinding, 0.358; filtering, 0.23. Total cost, including crushing and treatment, approximately, $3.50 per ton.

North Star.—

Crushing and concentrating,	$0.45
Cyaniding,	0.40
Total,	$0.85

I am indebted for much of the data to A. D. Foote, of the North Star mill; A. G. Kirby, of the Combination mill; Bernard MacDonald, of the Guanajuato Consolidated mill; C. W, Merrill, of the Homestake mill; Walter Neal, of Dos Estrellas mill; A. R. Parsons, of the Desert mill; and G. H. Rotherham, of the Montana-Tonopah mill.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes.

POSTSCRIPT.

BY H. O. HOFMAN AND W. MOSTOWITSCH, BOSTON, MASS.

In our investigation of the Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes,[1] we incidentally studied the decomposition of ferric oxide when heated in a current of dry air. We found that at 1,500° C., with 15-min. periods of heating, Fe_2O_3 hardly changed in weight, the greatest loss being 0.23 per cent., and we therefore concluded that the compound remained unchanged at 1,500° C. in a current of dry air. In a more recent experiment we heated Fe_2O_3 in a current of dry air and found that there was a decided loss in weight. A new investigation upon the behavior of Fe_2O_3, using a larger amount of substance and heating for a longer period of time, revealed the fact that a measurable dissociation took place at 1,375° C. The experimental data are given in Table I.

TABLE I.—*Decomposition of Ferric Oxide by Heating in Dry Air.*

Sample No.	Weight of Substance After Heating at 1,000° C.	Oxygen in Substance.	Temperature.	Time.	Loss in Oxygen.		Remarks.
	Grams.	Grams.	Deg. C.	Min.	Mg.	Per Ct.	
12	0.4584	0.1377	1,000	15	none.	none.	No change.
	0.4584	0.1377	1,100	15	none.	none.	No change.
	0.4584	0.1377	1,200	15	none.	none.	No change.
	0.4584	0.1377	1,300	15	0.6	0.43	Sintered, not magnetic.
	0.4584	0.1377	1.400	15	7.6	5.52	Sintered, magnetic.
13	0.4438	0.1333	1,100	40	none.	none.	No change.
	0.4438	0.1333	1,200	15	none.	none.	No change.
	0.4438	0.1333	1,300	30	none.	none.	Sintered, not magnetic.
	0.4438	0.1333	1,400	15	6.6	5.0	Sintered, magnetic.
	0.4438	0.1333	1,400	15	7.2	5.4	Sintered, magnetic.
	0.4438	0.1333	1,400	15	8.2	6.2	Sintered, magnetic.
14	0.3866	0.11615	1,100	25	none.	none.	No change.
	0.3866	0.11615	1,350	15	0.4	0.34	Sintered, not magnetic.
	0.3866	0.11615	1,350	15	0.2	0.17	Sintered, not magnetic.
15	0.2738	0.08226	1,100	20	none.	none.	No change.
	0.2738	0.08226	1,360	30	none.	none.	Sintered, not magnetic.
	0.2738	0.08226	1,375	15	3.6	4.4	Sintered, magnetic.
	0.2738	0.08226	1,375	15	4.2	5.1	Sintered, magnetic.

[1] *Trans.*, xxxix., 628 to 653 (1909).

The results in Table I. show that Fe_2O_3 heated to 1,360° C. in a current of dry air at atmospheric pressure for 30 min. remains chemically unchanged, and that at 1,375° C. the loss in oxygen amounts to 4.4 per cent. The substance which at 1,360° C. has sintered but was not attracted by the magnet, becomes magnetic at 1,375° C. and contains some FeO.

The results of our new experiments agree with those of P. T. Walden,[2] who upon heating Fe_2O_3 in an evacuated glass tube found that the pressure of the oxygen liberated rose from 5 mm. at 1,100° C. to 166 mm. at 1,350° C. Since this pressure corresponds to about one-fifth of an atmosphere, or to the partial pressure of the oxygen in the air, he concluded that Fe_2O_3 was stable in air to approximately 1,350° C.

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[2] *Journal of the American Chemical Society*, vol. xxx., No. 9, pp. 1350 to 1355 (Sept., 1908).

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

A Sea-Level Canal at Panama—A Study of Its Desirability and Feasibility.

Discussion of the paper of Mr. Granger, presented at the New Haven meeting, February, 1909, and published in *Bulletin* No. 25, January, 1909, pp. 1 to 37.

LEWIS M. HAUPT, Philadelphia, Pa. (communication to the Secretary*):—This subject is of great interest to me, and I am noting progress and conditions as a test of my own judgment in advocating what I believed to be the nearer, cheaper and better route at Nicaraugua. I have as yet learned nothing to change my views, or modify the convictions which led me to retire from the Commission; but that is not the issue to-day. Our government is committed to the Panama route, and is bound to "make good."

There are many points presented by Mr. Granger which seem to me worthy of very serious consideration. But when all is said there remains the risk of seismic convulsions, to which either form of canal is subject; and I fail to see that the risk would be reduced by making the cut deeper, with greater surcharge of lateral pressure upon the side walls. A severe earthquake in the rainy season would doubtless close either type of canal.

One of the arguments in favor of the Panama route most insisted on, was that Nature had provided good natural harbors at its termini. Recalling this, it has interested me to note that the cost of creating the necessary harbors is estimated at $27,000,000. Mr. Granger refers to this, and quotes the Board's report to the effect that, on the Atlantic side, it will require 5 miles of jetties to "make an excellent harbor out of a wretched one."

The vulnerable part of Mr. Granger's paper seems to me to lie in the admission that the plant required is still in the embryonic stage, and that at least one machine should be built and tested "to demonstrate that the theoretical strains and forces work out in practice exactly as expected." This, how-

* Received Dec. 19, 1908.

ever, is the usual process of evolution, and is the one followed by every conservative investigator. The system he advocates reminds me in a general way of one designed by my father for the Hoosac tunnel, and described in his paper on "Tunneling by Machinery,"[1] for which he received the Watt medal.

A. WOODROFFE MANTON, Murray Hill, Long Island, N. Y. (communication to the Secretary*):—The subject of Mr. Granger's paper is one concerning which I hesitate to speak confidently, but it seems to me that the possible damage to locks by earthquake, by ship-collision and from other causes, must have been very fully dealt with by engineers on both sides of the controversy. However this may be, it is to be regretted that a national undertaking of this character should be under any limitations.

I was fortunate in spending five years on the construction of the Manchester ship-canal in England, and I believe that the questions which arose in designing it were much the same as those which are now debated in connection with the Panama canal.

If the work can be so arranged that the sea-level prism can be excavated economically and without serious interference with traffic, when the need for it shall be demonstrated, the objections to the present plan would be to a great extent removed. Such a procedure would have the advantage of insuring better support to "soft" slopes than if the whole depth were now excavated "in the dry."

The present rate of progress is so admirable that it would seem a mistake, practically, to abandon the plan in hand and undertake a new one, involving many grave experiments, the results of which cannot be predicted with confidence. Mr. Granger's tunneling-machine displays great ingenuity, but in practice may not accomplish all that is expected of it. A heading machine can scarcely be considered in relation to, or compared with, a Lobnitz *dérocheur*. The latter does its work by making a face to break away to. It has only one plane to break up, so to speak, and that must be kept clear by dredging. The machine described by Mr. Granger has five faces

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[1] *Tunneling*, by Henry S. Drinker, p. 159 (1878). * Received Jan. 6, 1909.

and four angles to maintain, and as a working clearance cannot be preserved, these five surfaces can be advanced only by violently abrading, or pulverizing, the material under adverse conditions. These conditions are somewhat better towards the center, but are very bad along the perimeter, there being no constantly-maintained horizontal set-off to break from. I also fear that this difficulty would lead to a sloping hollow conical surface, with least radius at center, to work on the inside of which would distress the machine and tend to render effective blows difficult to deliver. To meet this, I would suggest a much larger ram, hollow, with annular cross-section, projectile-shaped; also, making the striking edge oblique, say from 10° to 15° to center-line of ram, so that when rotated suitably by a separate power-cylinder, and with the longest edge furthest from center of drift, it will be the better able to bite into the hollow cone which the machine may tend to evolve. I am inclined to think that the stresses set up with such a head would be less injurious and the attack more effective than with the glancing blow which a flat head must deliver on a sloping surface. The head should probably be heavy, and the stroke comparatively short, to render the ram as stiff as possible. But, by Mr. Granger's machine, would the rock be really broken away, as by the Lobnitz tool? If not, its output will be much smaller.

In considering Mr. Granger's very ingenious hydraulic shovel, it should be remembered that the invert will consist of a series of points, or teeth, which will tend to interfere with "bottoming-up," and the "mucking-out" so necessary to keep the face going uniformly ahead; this may be difficult at anything like the expected speed of operation.

Mr. Granger's filler-and-waster seems to be of excellent design, especially with unstable material, a bad dumping-ground, or both. It has the advantage of permitting the dumping of a whole train of cars at once, and there is no heavy loading of the bank-head, as with the cantilever design.

As to the flumes suggested, I should expect a considerable erosion, which would call for very heavy lining-plates, especially for sand or rock. I have seen chute-plates on a elevator dredge, 1.5 in. thick, cut away so rapidly as to need renewal in three months.

Capt. John C. Oakes,* Galveston, Texas (communication to the Secretary†):—In the discussion of the paper under consideration it seems advisable to consider the points accentuated by Mr. Granger before taking up the general subject of the abandonment of the present plan of the lock-canal, and the adoption of that of a sea-level canal.

I. Objections to the Lock-Canal.

The possibility of damage to a lock-canal by earthquakes is the first reason advanced by the author to support his contention. This subject was fully considered in all its aspects by the Board of Consulting Engineers. It cannot, of course, be said that the Canal Zone will be free from earthquakes, but I do not see that they would cause more damage to a lock-canal than to a sea-level canal. Any earthquake that would be severe enough to rupture the Gatun dam and allow the water to destroy Colon and the intervening valley, would probably cause the destruction of the city of Colon without any flood. It is known that much difficulty has been, and will be, caused by land-slides even with the 85-ft. level canal, and much more difficulty would be encountered if the excavation were made 85 ft. deeper. If the canal were excavated to sea-level an earthquake might cause the sliding of millions of yards of material into it, which would dam the Chagres and create a lake which, when it broke through, would have much the same effect as a break in the Gatun dam. All engineering works must be planned upon principles of safety under conditions that may be expected, and not under conditions that may be imagined. If all conditions that could be imagined must be provided for San Francisco would not have been rebuilt, nor would Galveston have been protected and raised, as has been done during the past few years.

The military objections to the lock-canal, it seems to me, may be disregarded. Either type of canal could be blocked by the sinking of a vessel in the channel, and that would probably be the easiest method of putting it out of commission for a short time. Danger to the locks from organized military operations can be foreseen and provided for. Some damage might, of course, be caused by an unauthorized or secret

* Captain, Corps of Engineers, U. S. Army.

† Received Feb. 3, 1909.

agent operating to destroy the lock-walls or gates. A small party, however, could do very little damage in the time that would be at their disposal, and by providing a proper guard danger from such cause could probably be prevented. As to explosives dropped from air-ships, it would require a greater stretch of imagination than I possess to believe that nitro-glycerine or other explosives could be carried over a lock-site in sufficient quantity, and dropped with sufficient accuracy, to do any material damage.

A much graver consideration is the liability of the locks to accident. Locks have been injured and gates carried away by the failure of signals to work, or, more often, by failure of the engineer to obey the signal. It is not uncommon for an engineer to go ahead when, according to signal, he should go astern, or *vice versâ*. The question of accidents to the locks, however, has received full consideration. Guard-gates are to be provided where necessary, and also movable bridges that may be swung across the heads of the upper locks to shut off the flow of water in case of accident. With these safeguards, and the proper regulation of the passage of ships, particularly by requiring that the propeller be not turned over at all while in the locks, there should be a minimum of accidents. The adopted plans contemplate the handling of all vessels in the locks by the electric mule.

Referring to fogs and tropical winds, I cannot see that the danger would be very much greater in one case than in the other, and in my opinion accidents from such cause will be rare in the canal, whichever type be adopted.

I doubt if there is any great danger from tropical burrowers. The dams will be of such enormous size and cross-section that it would seem impossible for burrowers to penetrate them. Moreover, the sea-level canal would require numerous dams, some of them large, as at Gamboa, and some to be earth-filled, for the purpose of controlling the water of the Chagres and contributing streams, and therefore the same liability to accident from this cause would exist in the case of either plan. The damage would of course be greater in the case of the destruction of the Gatun dam than of a smaller one, but the likelihood would be less.

Gen. H. L. Abbot studied the question of the water-supply

for a lock-canal very carefully. The result of his examination has already been published; [2] also the text of his paper.[3] It is shown that, by allowing a 4-ft. variation of level in the Gatun lake, an available water-supply of 12,270,000,000 cu. ft. would be carried through the dry season. This would provide for 1,577 cu. ft. per second for 90 days, which, added to the inflow, would make 2,577 cu. ft. per second. After allowing for evaporation, leakage, infiltration, lighting, power, etc., it is shown that an available supply of 1,350 cu. ft. per second would remain for lock-purposes, with a resulting conclusion that the available supply in the driest season would provide for 26 lockages at each lock per day during the three driest months. There are also pointed out available means for providing additional water-supply if the commerce passing through the canal should require it. There was little criticism of this conclusion by the Board of Consulting Engineers, and it would seem that the subject of available water-supply need cause no anxiety.

The only valid reason that has been advanced for a reconsideration of the plans for the canal has been the uncertainty of the foundations at Gatun. The statement of Mr. Granger, under the heading, "Uncertainty of Foundations," is, I think, fair; and if a sea-level canal of equal capacity and facility of traversing could be constructed for the same cost as the lock-canal, or less, there would be no room for argument.

Some of the most respected engineers in this country have at various times investigated and reported favorably upon the Gatun foundations. Recently Maj. William L. Sibert, Corps of Engineers, U. S. Army, member of the Commission and head of the Department of Lock and Dam Construction, after describing the various strata lying under the Gatun lock foundations, says:[4]

"The effect of pumping out the test-pit on the water-levels in the surrounding holes indicates, to my mind, that the water passed through small crevices in the rocks, rather than generally through the material, and that since these crevices are due to volcanic action the depth of them cannot be determined. This state of facts was unknown before, and while it has delayed the design of the locks, it does not impose, it is thought, any condition that cannot be fully met in the design."

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[2] *Report of the Board of Consulting Engineers for the Panama Canal*, pp. 74 to 76.

[3] *Ibid.*, Appendix E.

[4] *Annual Report of the Isthmian Canal Commission for* 1908.

Maj. Chester Harding, Corps of Engineers, U. S. Army, Assistant Engineer on Foundation of the Gatun Locks, reports:[5]

"There is no doubt of the ability of these different materials to bear the greatest load that will be transmitted to them by the lock-walls, and the locks may be safely founded upon them if means are taken to exclude the underground flow of water from the softer materials on which some of the walls will rest."

C. M. Saville, Assistant Engineer in the Gatun dam investigations, reports as a result of his studies during the past year:

"1. That suitable material is available and near at hand for the construction of the Gatun dam by hydraulic process.

"2. That the foundations are suitable for such a structure as the proposed Gatun dam if they are properly treated.

"3. That it is practically possible to construct a stable and water-tight earth dam at Gatun of the material available.

"4. That the hydraulic method of construction, as proposed for this work, is feasible if proper conditions are observed."

These reports, which I suppose are approved by the Isthmian Canal Commission, should remove any fear that the construction of the Gatun dam and locks may not be carried to a successful issue.

II. The Proposed Method.

Mr. Granger says that, as the excavation for the lock-canal could perhaps be completed in three years, it might seem unnecessary to suggest a method for more rapid and less costly excavation. Then, assuming that the objections raised to the lock-canal are sufficient to require a change of the plans, if better methods should be provided for the excavation of a sea-level canal, he takes up the question of such methods and proposes a scheme novel in conception and, I believe, absolutely untried on a large scale up to the present time. He furthermore says that his plan would require an expensive installation, which, if assessed upon subsequent work, would amount to only a small sum per cubic yard, a large part of which could afterwards be recovered by the sale of the apparatus for use on other engineering works; and, finally, the whole amount would be fully recovered by the economy in money and time secured by such an outlay at the beginning.

It will readily be conceded that the abandonment of present

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[5] *Ibid.*, Appendix D.

methods and plant, the changing of plans and construction of new plant and auxiliary works would be expensive, but it is not clear that a saving in either time or money would result.

The machines recommended are described with so little detail that it is difficult to form a definite opinion of the proposed plant and methods, but, if I understand the text of the paper, the following remarks express my views of the machines:

The grading and pile-driving machine, as described, is not new, except possibly in some of its details. Machines of this kind have been used on other works, although possibly not combining so many different functions. In my opinion, it would be impossible to work such a machine economically. The orange-peel bucket would be a very inefficient tool for handling the material found in the canal prism; certainly, a machine moving on a track and carrying an orange-peel bucket, leads for pile-driving, air-compressors, drills for blasting, and arrangements for substituting a shovel for the orange-peel bucket, would make a very cumbersome and awkward machine, less efficient than separate machines for such work. On the Aransas Pass and Galveston jetty work we have found a pile-driver with air-compressor for boring, etc., and a separate hoister for placing material, much more efficient than one combination machine could possibly be.

The "hydraulic river and harbor dredge" seems not to differ from other hydraulic dredges except in the size of its pump, the strength of its machinery, and the addition of a clam-shell or orange-peel bucket, to be operated from the bow. The objection to a combination machine must again be made to this suggestion. To attempt to operate a clam-shell or orange-peel bucket and suction-dredging machinery from the same float would, in my opinion, cause a loss of time and result in inefficiency. Neither the clam-shell nor the hydraulic machinery could be worked to full capacity. The stoppage of a dredge of the capacity suggested would entail very heavy expense, materially affecting the cost per cubic yard excavated. Nor is it clear why electricity should be suggested as motive-power. This subject has received careful attention by the canal engineers, and they have decided to use steam, with oil as fuel. It must be supposed that if electricity were more economical it would have been adopted.

The details of the suggested "method for making large railway-fills without the use of expensive temporary trestles" are not given with sufficient exactness to permit detailed criticism. In my judgment, however, the construction and maintenance of a conveying-belt, such as is described, and of the length which I imagine is intended, with the conveying-machinery, towers, cables, etc., would result in many complications and loss of time and money. The statement is made:

"It is perfectly practicable to design an apparatus of this kind which will deposit more than 10,000,000 cu. yd. without resetting, at a cost, for the plant, of about 3 cents per yard of the first dump, an operating cost of not over 2 cents per yard (including generation and transmission of power), and a capacity of 60 cu. yd. per minute."

I do not understand how 10,000,000 cu. yd. of material could be deposited anywhere along the line of the canal without resetting the machine. Certainly, the delivery of 60 cu. yd. per minute, or 3,600 cu. yd. per hour, or 28,800 cu. yd. per day of 8 hr., presupposes remarkable digging and supplying of material to this machine. To supply material to keep such a machine running to the stated capacity would require over 1,500 cars per day unloaded on to the belt, or more than three per minute, without allowing for switching or any other stops. Such delivery appears to me impossible; and the capacity of such unloader, even if a practical machine, seems overstated. One of the great difficulties in digging the canal is that of finding dumping-places for the material excavated, and to think of depositing 10,000,000 cu. yd. of material at any one place on the Isthmus without resetting the dumping-machine seems to me to show a lack of understanding of existing conditions.

Mr. Granger next suggests "a design whereby a full train of cars could be loaded in a cut or tunnel behind one or two steam-shovels." I cannot see any benefit to be derived from this method, since it supposes the digging to be done by a shovel which delivers to the belt, the belt delivering to a chute, which delivers to the cars. It seems to me that this method complicates the ordinary method of depositing directly into the cars, and offers more opportunity for stoppage of the shovel, owing to lack of co-ordination and breakage of the different parts, besides the added difficulties of moving apparatus ahead and a greater installation-cost.

Of the hydraulic shovel I do not feel that my experience entitles me to speak.

Mr. Granger next suggests "a movable cantilever for dumping one or more cars from the end of a track." If I understand the plan of the device as shown, the cantilever is supported on a length of track equal to that covered by three cars.

No details are given as to the size and weight of this cantilever, and I have not had time to make a careful estimate of its weight; but according to a rough estimate prepared in my office this apparatus, as shown in Fig. 2, would weigh 150 tons without any balancing-weight, if designed to carry three 50-ton flat-cars on the overhang, as shown on the drawing. These cars, many of which are now used in transporting excavated material at the Isthmus, weigh from 16 to 18 tons each, and each will carry from 24 to 30 tons of material. Each loaded car will, therefore, weigh at least 40 tons; assuming 6 loaded cars on the cantilever, as shown on the drawing, they would weigh 240 tons. With the truss as shown, which for all practical purposes may be assumed to be symmetrical, all of this weight would be concentrated under the upright at the center of the truss. In addition, there must be a balancing-weight on that end of the truss resting on the track, part of which will be transmitted to the same point, making a concentrated load under the center of the truss and at the end of the track of at least 400 tons. If we assume that Mr. Granger intends this cantilever to have only one-half this capacity, there would be a concentrated load at the end of the track of approximately 200 tons. The track must be pushed out as the dump is extended, and it must of course be laid on unsettled material, with consequent sinking and shifting of track, and, in my opinion, the weight of this cantilever absolutely precludes its use as proposed. Much of the difficulty on the canal has arisen from the shifting of the tracks under the loaded trains, both in the cut and on the dumps, particularly during the wet season. Whole trains of cars have been derailed from sliding tracks. How much more would this be the case if a load of from 200 to 400 tons be applied to the newest and unsettled portion of the embankment.

Mr. Granger next describes a rock-mining and tunneling machine. As I have had practically no experience in mining and tunneling, I will refrain from any criticism of this device.

Mr. Granger presents a table taken from the report of the Board of Consulting Engineers, showing the quantity of material to be excavated for a sea-level canal at the date of the report (January, 1906), to be 231,026,277 cu. yd., and, in Fig. 5, attempts to show that to widen the canal from 200 ft. to 300 ft. would require the excavation of an additional prism of one-quarter the original.

The quantity given above was not for a 200-ft. canal. The proposed canal to which the figures refer is as follows:[6]

"Summarized, the sea-level canal as recommended by this Board is a channel commencing at the 41-foot contour in Limon Bay, about 5,000 feet northerly of a line between Toro and Mazanillo lights, protected by two diverging jetties with a width of opening of 1,000 feet; thence with a straight channel 500 feet in width at the bottom and a depth of 40 feet protected by a parallel jetty on the west and by Manzanillo Island on the east, to Mindi, where the land canal begins.

"This canal is designed with a depth of 40 feet and a bottom width of 150 ft. in earth, with side slopes adjusted to the nature of the ground, so as to give a surface width of from 302 feet to 437 feet. In rock the section is to be altered so as to have a bottom width of 200 feet and a surface width of 208 feet. At the Pacific end the canal is to be protected by a tidal-lock located between Ancon and Sosa hills. Beyond this tidal-lock there is to be a straight channel projected into Panama Bay, with a bottom width of 300 feet and extending for a distance of three and three-fourths miles to the 45-foot contour."

Over 41 per cent. of the length of this canal was to be only 150 ft. wide on the bottom.

In February, 1904, M. Bertoncini, chief draftsman on the Isthmus, submitted to the Chief Engineer, Mr. Wallace, an estimate of 250,000,000 cu. yd. excavation for a canal 200 ft. bottom-width between Bohio and Miraflores (miles, 17.22 to 40.77). To this the quantities for the stretch between Mindi and Bohio (miles, 5.49 to 17.22) and between Miraflores and Sosa Locks (miles, 40.77 to 45.37) must be increased approximately 25 per cent. to give a fair estimate of the excavation for a canal with 200 ft. bottom-width. The total would be, approximately:

	Cu. Yd.
A,	2,922,734
B,	10,047,473
C, increased 25 per cent.,	30,696,730
D, E, F, Bohio to Miraflores,	250,000,000
G, Miraflores to Sosa Locks, increased 25 per cent.,	14,163,160
H, Sosa Locks to 45-ft. contour Pacific,	6,172,867
Total, approximately,	314,002,964

[6] *Report of the Board of Civil Engineers*, p. 50.

This shows that the figures given must be increased about 36 per cent. for a canal 200 ft. wide on bottom, and this increase is mostly in material which is excavated at the maximum cost, a large percentage of it being rock.

If it requires an increased excavation of 83,000,000 cu. yd. to widen the canal from 150 to 200 ft., it should require at least three times that amount, or 249,000,000 cu. yd., to widen the canal to 300 ft., or approximately a total of 480,000,000 cu. yd. based on estimates made previous to January, 1906. The amount of excavation required to transform the proposed lock-canal to a sea-level canal with 300 ft. bottom-width was estimated[7] at 241,900,000 for the stretch between Gatun and Kilometer 59, which is near Pedro Miguel, which, added to the amount required to be excavated for the 85-ft. level canal between the same points, 60,200,000, gave 302,100,00 cu. yd. for the amount to be excavated for the sea-level canal, 300 ft. wide, between Gatun and Pedro Miguel. This estimate does not include the excavation from the sea to Gatun, nor that from Pedro Miguel to the Pacific, for which quantities I am unable to find any accurate figures; but estimating from all the data at hand, I place the amount at about 80,000,000 cu. yd., which would make a total of approximately 380,000,000 cu. yd. The figures given[8] were obtained by increasing the quantities for the sea-level canal by 70 per cent., and I believe the result is much too small. The total yardage for the canal 300 ft. wide would, of course, depend on the slopes, and, in my opinion, would undoubtedly be between 400,000,000 and 500,000,000 cu. yd. This great increase as the bottom of the canal is widened is due to the narrowness of the valley in which the canal must be excavated. The hills adjacent to the canal through the big cut would in some cases have to be sloped back almost to their tops. In other words, the sides of the hills are so close that in the case of a 300-ft. canal a one-on-one slope, at points along the line, would cut into the hills and come out at their highest points. A few illustrations have been published[9] which show how the hills would have to be cut away.

[7] *Report of the Board of Consulting Engineers, Appendix P.*

[8] *Ibid.*, Appendix P.

[9] *Engineering Record*, vol. lviii., No. 10, pp. 256 to 261 (Sept. 5, 1908); and *Report of the Isthmian Canal Commission for* 1908 (plates 3, 6, 11, 17, etc.).

Mr. Granger's estimate of the amount of excavation necessary for a canal 300 ft. wide on bottom is therefore vitally defective.

Mr. Granger apparently would allow the Chagres to flow into his proposed sea-level canal practically without regulation, by providing only a stepped apron a mile long. He thinks the current developed would be no real obstacle to navigation, and compares the navigation of the Magdalena and Atrato rivers to that of the proposed canal. I would ask if large ocean-steamships with their box cross-section and small rudder are used on those streams. Judging from my experience, such vessels are often unmanageable in contracted water-ways with a tidal current of even 1 or 2 miles per hour. In my opinion, no master would risk his vessel in a 300-ft. channel with a current of 5 miles per hour. Mr. Granger also thinks that only a small amount of dredging would be necessary at the point where the Chagres enters the canal. I venture the statement that if the Chagres in flood were allowed to enter the canal, even if stepped down over an apron, the banks and bottom of the canal would be eroded and shoals and bends formed from the point of entrance to the sea, except where the cut was through rock.

Erosion, shoaling, and difficulty of navigation apply also to the proposed omission of the Pacific tide-locks. I am certain that a velocity of 4.9 miles per hour at the entrance of the canal would prohibit the entrance of ordinary ocean-going vessels. The conditions in the Mississippi at New Orleans, and in New York harbor, are not comparable to the conditions that would exist at the Pacific entrance of a sea-level canal without a tide-lock. While I am not familiar with the conditions at Cartagena and Buenaventura, or on the rivers Anchicaya, Cajambre, or San Juan, I believe an examination would show that they cannot properly be compared with the conditions that would exist in the canal. It is not alone the velocity of water in a channel that determines the question of its navigability by ocean-steamers, but the width and depth of channel are vitally important. With the same current, ease and safety of navigation may be approximately measured by the ratio of areas of cross-sections of the channels.

Many accidents to vessels in contracted channels could be

cited, but I think it unnecessary. Most people who have had to do with shipping know the danger to large ocean-going ships under such circumstances.

With reference to the subject of jetties, Mr. Granger says that his plan will require 5 miles of jetties on the Atlantic side, reaching to the 41-ft. contour, and 5 miles on the Pacific side, reaching to the 45-ft. contour. He then assumes a cross-section 21 ft. wide on top, with an average depth of 51 ft., with side-slopes of one-on-one and one-half, which would require 9,724,000 cu. yd., and estimates the cost at 30 cents per cubic yard. It would perhaps be possible to build a rubble-mound jetty with the assumed slopes, but not for 30 cents per cubic yard. In my opinion, this estimate is very much too low. The rock excavated from the canal prism is generally an indurated clay, with some volcanic dikes of hard material penetrating it, and not suitable for jetty-construction. It would be necessary to select the material at increased cost or to open quarries for the purpose of providing it. The latter method would probably be adopted. These quarries would have to be at a considerable distance from the jetties on the Atlantic side, but might be found close to the shore on the Pacific side. The slopes of the jetties must be covered with blocks weighing from 3 to 20 tons from the bottom upward, and the cap-blocks must be specially quarried rock of at least from 10 to 20 tons' weight, the sides generally rectangular in shape, so that they may be fitted closely together. The quarrying on the Isthmus cannot be done for less than 50 cents per cubic yard; the transportation and placing in jetties, no matter what method is used, will certainly cost from 20 to 50 cents per cubic yard, and considering the cost of new plant, quarrying-apparatus, trestles and mattresses if used, locomotives, cars, etc., and allowing for deterioration, I cannot see how it is possible that these jetties can be constructed at a cost of less than $1 per cubic yard. The cost of quarrying, putting rock on cars, and placing in the jetties at Aransas pass, Brazos river, and Galveston, amounts to from 75 to 95 cents per ton, without considering contractor's profit, hauling, new plant, or deterioration of plant. Assuming, therefore, that Mr. Granger's cross-section is correct, it will be seen that the cost of the jetties would be nearly $10,000,000, instead of the amount, approximately $3,000,000, as estimated by him.

I have already shown that the cantilever proposed for use at the end of the dump could not be used on earth-dumps. If used for jetty-building it must be carried out on a trestle or on the jetty. The cost of a trestle to support the cantilever safely would be prohibitive where the depth of water was over 20 ft., and if the track be placed directly on the jetty the latter must be leveled and prepared for the tracks as often as the jetty is extended the length of the overhang. During the time that this work is being done all dumping-operations must cease; moreover, the immense weight of this cantilever, concentrated practically at the end of the newly-completed mound, would undoubtedly cause the material to push outward in front and on both sides, particularly at Colon, where the bottom is soft. I do not believe that this cantilever is a practical machine for jetty-building.

Mr. Granger proposes, if I understand him, to handle most of the material excavated from the canal prism, particularly that in sections C, D, E, and F, of Table II. of the paper, by pumping the excavated materials into a flume, which is to be 150 ft. above mean sea-level, and then to provide sufficient water to the flume to transport this material to the Pacific slope. I have not the data at hand to enable me to check his estimate of the cost, but upon several points he is not clear. He does not show where he will obtain the water-supply to keep the flumes in operation, either at Gorgona or Obispo, and I am not sure that such supply is available. Even if it is available, large pumping-plants must be installed. He expects to be able to handle all kinds of material, including rocks weighing, I judge, as much as 500 lb. It would not be impossible to give sufficient slope to the flumes to carry the material, but if the flow were stopped, due to breakage of pumping-plant or flume, it seems probable that a dam would be formed in the flume which, when the water was turned on again, might cause overflow and great difficulty in resuming operations. As much of the material to be moved is clay, it is probable that difficulty would be caused by its sticking to the bottom of the flume, with the result that a shoal would be formed that would ultimately dam the flume. His scheme also includes the rehandling of the material carried by the small flumes into the larger flume. No estimate is included for plant for

water-supply or for rehandling material from small to large flume.

Mr. Granger seems to contemplate also the removal of all the steam-shovels, tracks, etc., in the great cut and the substitution of powerful dredges to pump the material to the flume. In my opinion, this method is not feasible. He has made no estimate for the auxiliary works, such as dams, spill-ways, aprons, etc., for the controlling of the Chagres and the 20 or more contributing streams, which would have to be let into the canal, diversion-channels supplied or the flow turned away from the canal, all of which would necessitate the construction of numerous dams, spill-ways, and channels.

I think that it has already been shown that Mr. Granger's estimate of the amount of material to be excavated and his unit-prices are underestimated, and I do not believe that the machines designed and suggested for use on this great work are any improvement over those in use to-day, or would effect any decrease in time or cost of the completion of the canal.

The type of canal that will finally be constructed will in all probability be a lock-canal, unless it be found that the foundations for the dam and locks at Gatun are not suitable for those great structures. Each type of canal has its special advantages and disadvantages. One of the great difficulties in providing a sea-level canal is the controlling of the Chagres and all of the streams entering thereinto between Gamboa and Mindi; and this difficulty, with the added consideration of better facilities for navigation offered by the lock-canal, was, I believe, the principal reason for the minority recommendation for a lock-canal. This type will give wider channels and less dangerous navigation than the sea-level canal except at those points where the vessels must pass through the locks, and for the 9 miles in the section known as the Culebra cut, which must be the same for both types. On the other hand, the locks are subject to damage from various causes, but such causes can be foreseen and measures provided to minimize such damage.

In my opinion, a lock-canal with an elevation of 60 ft. would have combined a greater number of advantages of both types with a less number of disadvantages than the 85-ft. level canal authorized by Congress. A reduction of the proposed level by 25 ft. would bring the time of completion of the excavation

nearer to that for the completion of the masonry, and would not increase the cost of the canal probably more than 15 or 20 per cent. This opinion is not original with me, for a canal at the 60-ft. level has been studied for many years, and was the type adopted by the Board of Consulting Engineers by a vote of eight to five as being the best type of lock-canal to study for purposes of comparison with the sea-level canal. One of the members of the Board of Consulting Engineers who signed the minority report, General Abbot, voted for the 60-ft. level as opposed to the 85-ft. level, but later voted for the 85-ft. level as opposed to the sea-level. It may be that this subject will be reopened because the foundations at Gatun may be found amply secure for two locks, while not sufficient to warrant the construction of three locks, and for this reason the level may be lowered from 85 to 60 ft.

A lock-canal, however, at either the 60- or the 85-ft. level is superior for navigation-purposes to any sea-level canal that could possibly be constructed at anything like equal cost.

R. R. HANCOCK, New York, N. Y. (communication to the Secretary*):—Like most other Americans, I had regarded the Panama question as settled in favor of the lock system. Now, however, in view of the facts and arguments brought out in Mr. Granger's paper, I am convinced that a sea-level canal is the only one that should be built. Without laying stress on the minor arguments, it seems to me that the undeniable dangers that would exist in the use of a lock-canal at Panama are:

1. Insecurity of locks, however massive, in an earthquake-region.

2. Liability to an insufficient water-supply, possible in such a pervious formation, where the filtration and leakage may be higher than that indicated by Mr. Granger.

3. Lack of proper and impervious dam-foundations.

From all of these dangers, as well as from high annual maintenance-charge, and the risk of all accidents that might seriously interrupt traffic, the sea-level canal would be free.

The facts brought out in Mr. Granger's paper indicate that even under the present methods of construction a sea-level canal, according to the original plan with 200 ft. bottom-width,

* Received Jan. 26, 1909.

could probably be finished in approximately the same time as a lock-canal. With such rapid construction-methods as proposed, and the hydraulic transportation of the material, the cardinal points of which are indorsed by so many engineers of repute, showing a method whereby apparently in perhaps less than four years' time a canal at sea-level could be finished with a bottom-width of 300 ft. in the clear (this 50 per cent. increase in width eliminating all the difficulties of the original plans) at materially less cost than anticipated for the completion even of the lock-system, and leaving on hand a splendid equipment ready for work at home, it seems incumbent upon Congress to take thoughtful action at an early date upon the matter.

GUSTAV H. SCHWAB, New York, N. Y. (communication to the Secretary*):—The engineers have debated concerning the type of interoceanic water-way to be constructed at Panama; but has the American man of business been asked for his opinion? He has listened to the engineers' discussions, he has heard why dams have been moved from one site to another where it is expected that they will be able to fulfill their purpose and to stand when built; and he has been told that it is now the final intention, trusting in the stability of these dams, to build a lock-canal, to convey from ocean to ocean, for all time to come, the commerce of the world.

The United States army engineers enjoy a deservedly high reputation. If they declare that they can build a dam at Panama on a foundation of clay or any other soft material, to stand for ages, the layman can express no dissenting opinion; but on the question of lock-canal *vs.* sea-level canal I believe that the ship-owner or commercial shipper has something to say, and should utter his opinion before the country is irrevocably committed to a definite plan of canal-construction. For it is he who is expected to use the canal when built; and he should demand that any canal built by the United States at Panama shall be the best that can be constructed. He knows that the lock-canal as proposed in that locality is an inferior type of canal and that a sea-level canal is the more serviceable and more efficient water-way, offering safe and uninterrupted naviga-

* Received Feb. 20, 1909.

tion for the commerce of the nations as long as the oceans exist. This is an indisputable fact, confirmed by the highest authority.

The first consideration for an interoceanic ship-canal is that it shall be entirely free from avoidable obstructions. An obstacle that would either delay or endanger the passage of vessels through such a water-way must be removed. Locks would present a constant menace to safe navigation.

In justification of the lock-canal at Panama frequent reference is made to the Sault Ste. Marie canal and to the Manchester ship-canal in England. On the Sault Ste. Marie canal within a period of 9 years there were three accidents involving injury to lock-gates, and on the Manchester ship-canal during the same period there were at least the same number of accidents to lock-gates. In the case of the Sault Ste. Marie canal there is only one lockage, with a lift of 20 ft. As six locks are now planned on the Panama canal, each with a lift of about 30 ft., it seems to the practical man probable that the number of accidents to be expected would naturally be multiplied by the number of locks. A further element of danger is presented by the construction of three locks in one flight, one communicating directly with the other. A steamer approaching the upper lock and passing beyond the control of her commander through a mistaken order, or an order wrongly executed in the engine-room, would pass through the upper gate; and, by doing so, open up the entire flight of locks to the flood from the lake above, which would not only engulf the steamer but would ruin the locks and their appurtenances, creating a situation that it would, no doubt, take a long time to set right, while navigation would be closed.

The traffic on the Sault Ste. Marie canal consists of regular lines of steamships, the same boats passing through the lock at frequent intervals and therefore conversant with conditions there. But where flights of locks of the enormous size planned at Panama will be used by the vessels of all nations, and of various sizes and descriptions, many of them at infrequent intervals, the possibility of accident in connection with these locks becomes, to a practical shipping-man, a vital question.

Mistakes in the transmission of orders from the bridge to the engine-room are fortunately not of frequent occurrence, but

when they do happen the result is often most disastrous. Experience has shown that steamers, when being docked at their piers, have, through such mistakes, run into heavy piers and quays, cutting into steel, wood, cement, and granite, even though under very little way. The large, high-power steamer of the present day weighs many thousand tons; and this weight, once in motion, even at the slightest speed, will pass through anything that is in its path.

Obstructions to a sea-level canal might be caused by the accident of a vessel stranding or sinking, but such obstructions could not be other than temporary, and could be removed in a few days. They have frequently occurred in the Suez canal, and have been removed either by blowing up or by other means, without causing great delay. Compared with such delay of a few days, the disastrous consequences of an accident to the lock-gates which would cause the most serious damage and drain the water out of the summit lake would be of an entirely different character.

The views here stated on the relative merits of a lock-canal and a sea-level canal represent the opinions of masters, mates, and pilots—of those men who are most competent to judge of the obstructive character of a lock-canal in the navigation of the present day. The American Association of Masters and Pilots of Steam-Vessels is on record as indorsing the sea-level canal rather than the lock-canal, owing to "the removal of the risk of damage to ship and cargo by the adoption of a sea-level type." The United New York-Sandy Hook Pilots' Association is also recorded as strongly and unhesitatingly advocating a sea-level canal at Panama, as it has been the experience of the Sandy Hook pilots in the hazardous occupation of handling large vessels "that there is always more or less danger of an order, correctly given by the pilot from the bridge, being misunderstood or misinterpreted and incorrectly executed." Such misunderstandings, the pilots say, "are always liable to be costly by causing damage both to pier or wharf in docking and undocking steamers, and also to the vessel being docked, in the waters of New York harbor, where there are no locks to contend with or to be damaged."

The fact that, as a supposed protection against the possibility of damage to lock-gates, it is now proposed to duplicate each

gate, shows that the authorities in charge of the construction of the canal at Panama fully realize the dangers that are involved in the use of a flight of locks, as proposed at Gatun, by full-power steamships of enormous size and weight.

The interference by earthquakes with the safe operation of a lock-canal for continuous and uninterrupted navigation from ocean to ocean, is another and serious objection. Omitting any reference to the frequency of earthquakes on the Isthmus in former years, the statement has been made, and I believe has not been refuted, that since 1901 and until 1905 there were seven tremors and four light shocks at Panama, the latter affecting buildings to an appreciable extent. What the consequences would be on locks built on earth-dams not founded upon rock cannot be forecasted.

The locks at Panama, originally planned for a width of 95 ft., have now been increased in width to 110 ft., with a length of 1,000 ft., to accommodate expected growth in the size of vessels. Thirty years ago no one in his senses would have dreamed of ocean steamships such as we now see in any large harbor of the world. Forty years ago the largest steamer measured 350 or 375 ft. in length, with a beam at the most of about 40 ft. To-day we have steamers 800 ft. long, of 80-ft. beam, and others projected of still larger dimensions. He would be a rash man who would predict what size the vessel of 50 years hence will have. This much is absolutely certain: the dimensions of vessels will increase, and harbors, channels, docks, and piers will be deepened, enlarged, lengthened, and widened to accommodate the inexorable demands of commerce and navigation. We may not live to see the dimensions of vessels increase very much beyond the present type, but the canal at Panama will live to see it; and, if circumscribed in its capacity by its locks, 50 years hence it will be a canal largely limited in its usefulness and distanced and outclassed by its rivals. The Suez canal will be an active competitor with the Panama canal for the trade to the East, and with a sea-level canal, such as exists at Suez, the deepening and widening is simply a question of a few millions and a few years. But a lock-canal of the dimensions of that at Panama will be precluded from enlargement by the enormous cost of reconstruction of locks on a larger basis, involving the closing of the canal for a number of years. It may

be said that this is a matter of the future; but in the construction of a work of such far-reaching and world-wide importance as the water-way at Panama, even remote questions that will affect our children should not be treated lightly.

Considerable weight is attached to the alleged greater speed of operation on a lock-canal as compared with a sea-level canal. This, it appears to a practical shipping-man, is illusory. Although the present plan involves a large extent of lake-navigation, the limits of the channel through the lakes would impose upon passing vessels an obligation to observe a slow rate of speed, so that it hardly appears probable that the lake-navigation would offer any advantage over the passage through an unobstructed sea-level canal, especially if the latter is finally transformed to approximate the type of a "strait."

If the lock-canal presents grave objections to commercial men, what may not be its effect upon the life of the nation in a time of war? The failure of a lock-canal at a supreme moment in such a war may be fatal. Should it be necessary to send a fleet of battleships from ocean to ocean at a critical period, what might not be the consequence if a lock-gate should be jammed or disabled or destroyed either by accident or hostile design during or before the passage of the fleet?

Although the cost of construction of a sea-level canal would, no doubt, exceed that of a lock-canal, the business interests of this country would do well to consider that the annual cost of maintenance and operation of a sea-level canal would necessarily be less.

Since the comparative merits of a sea-level canal and a lock-canal were last officially considered, three years ago, entirely new conditions have arisen. The estimated cost of the completion of a lock-canal is now double the sum then estimated; meanwhile, the improvements and inventions made in excavation in other parts of the world have shown that progress is very much greater and cost very much less in such excavation than was originally assumed. Both of these considerations are important in the construction of a sea-level canal. The great reduction in the cost of rock-removal and the accelerated rate of progress in all excavation-work would, no doubt, considerably shorten the time that was originally assumed necessary for the construction of a sea-level canal. Under these changed

conditions the construction of a sea-level canal appears possible from the stand-point of cost and time. The value of such a canal is so great that the question of $100,000,000 more and of a few years longer time should not be permitted to stand in the way.

The Board of Consulting Engineers, appointed by the President of the United States in 1905, submitted a plan for a lock-canal that could be transformed into a sea-level canal without interrupting the traffic upon it, but it favored the immediate construction of a sea-level canal in view of the cost and difficulty of such transformation. To what extent the remarkable reduction in cost of rock-excavation elsewhere and the rapid progress attained in such work would modify this opinion is a technical question to be solved by engineers.

The supreme importance of the canal through the Isthmus of Panama to this nation in peace and in war, and the appalling consequences that may result from a wrong decision as to the type of canal to be adopted, render it, in my opinion, imperative upon all who have the best interests of our country at heart to demand a reconsideration of the plan before it is too late.

H. L. MILLNER, Washington, D. C. (communication to the Secretary*):—When we consider that the Isthmian canal is being built for all time, and that national defense is a prime reason for its existence, minor questions of expediency should be eliminated. Assuming, as appears to be admitted, that either type is structurally feasible, the discussion should be narrowed to those features which involve permanency and simplicity of operation.

The inconvenience and cost of operating a series of locks of such magnitude as is necessary in that plan cannot be denied; and the grave questions of stability raised by eminent engineers remain as pertinent as ever. These appear to concern: (1) the stability of the Gatun dam; (2) the ability of the lake to maintain its level during the dry season; (3) the liability of serious damage or total destruction of the locks by earthquake, accidental ramming of the gates by vessels, or the work of public enemies.

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* Received Feb. 24, 1909, but proof not yet revised by the author.

Assuming that the Gatun dam can be made safe by suitable construction, the constant vulnerability of the locks still remains; and when due consideration is given to the far-reaching consequences of a protracted interruption of canal-traffic and the possibility of naval disaster which it might easily involve in time of war, these serious features cannot be set aside by the mere statement that earthquakes, accidents, and wars are not likely to happen.

Were there no other choice, the urgent demand for the interoceanic canal would justify the assumption of these risks, but we have as an alternative the sea-level canal, originally recommended, and even now admitted to be ultimately desirable.

In my opinion, it is doubtful whether, if constructed by up-to-date methods, the sea-level canal, closely approaching a work of nature, would cost materially more than the plan now in progress, and I am further of the opinion that tidal locks and the proposed Gamboa dam would be unnecessary with the sea-level type, since the currents in the canal caused by tidal difference and floods in the Chagres river would be very light in the first case (less than 5 miles per hour maximum at times), and in the second case could only result in temporary embarrassment at infrequent periods.

Should tidal locks and the Chagres dam be considered necessary, however, the balance of simplicity and safety is still in favor of the sea-level type.

W. L. Saunders, New York, N. Y.:—Mr. Granger's estimates of the cost in money and time of a sea-level canal at Panama are too largely hypothetical and general to prove that such a canal could be constructed as quickly and cheaply as he imagines; and his proposed method of handling the Chagres river seems to me highly problematical. In fact, the Chagres river, and not the mere cost of digging, is the chief question involved. The present plan for a lock-canal takes care of this element. Any other plan, however cheap or quick in construction, must do that, or it will be impracticable.

My business is rock-excavation, and the part of Mr. Granger's paper that interests me most is his proposed system of boring. On that point I have been requested by him to say something. His plan is only on paper, never having been put into practical

operation under conditions such as exist on the canal, and to say that his scheme is practicable is merely to pass judgment upon the working-value of a device which one has examined through the specifications and drawings of a patent. My experience with patented devices has led me to the conclusion that of the numerous, plausible ideas that go through the Patent Office not 1 per cent. ever materialize in practice.

Now, it seems to me beyond dispute that in a great work, like that of building a canal across the Isthmus of Panama, only such designs and apparatus should be employed as have been tried to the point of success elsewhere. This is not a case where the brains of able engineers should be employed to design machinery to do the work, but one where practical men should select such machinery as has been proved to be competent and economical when put to work under similar conditions. My friend, Mr. Granger, has produced some ingenious designs, for he is very fertile in ideas, and I am inclined to say, "Go ahead with your plans; they show ingenuity and ability. Get somebody to try them out somewhere over in New Jersey, and if they work all right, then come and let us talk about applying them to the Panama canal." This statement applies equally to Mr. Granger's device for handling material, and to his pounding-machine. He proposes to hit the rock with a sledge-hammer blow, which might be compared to a battering-ram of ancient times. The idea is new and ingenious, and he may be entitled to a patent. Personally, I do not think the machine will successfully ram rock under the conditions that exist on the canal. Mr. Granger has done me the honor to mention my name in his paper, and while I do not intend to bore you with an argument as to what I did or did not say to him, you will pardon me for imposing on you my personal opinion now in the statement that it is my belief that a machine such as he has designed may be able to work; that is, it will strike the rock and strike it hard, but it is designed on so large a scale—one involving such complications of apparatus, subject to breakage- and repair-costs—that I do not believe the machine to be a practical one; and even were it practical, that is, even though able to stand the hard service, it is not likely that it will succeed in battering a tunnel-hole in rock of moderate hardness.

The Greeks and Romans had battering-rams that probably struck a harder blow than any rock-drill of to-day, yet notwithstanding these rams walls were built around ancient cities until explosives came into use, which, when inserted in a small drill-hole, did far more damage than the ram. Mr. Granger's idea seems to me to be a step backward, because it is cheaper and more expeditious in removing rock to use explosives than to batter the material out by main force. This is especially true in tunnel-driving, and even in open work the drilling-and-blasting system is more expeditious and cheaper except in certain classes of subaqueous excavation, where thin layers of rock only are to be removed and where the material is favorably laid.

CHARLES WHITING BAKER, New York, N. Y.:—I do not wish to repeat what has been so well said by Captain Oakes, but, as I have just come from the Isthmus, some of the facts I learned there may be of interest.

To take up only two or three of the points that have been raised in this paper: The common idea of the layman is that a sea-level canal is a simple thing to build; that it is only so much digging and the canal is done. That idea is held by many capable engineers who have given only casual study to the matter. Curiously enough, their attention is all concentrated on the Culebra cut. Their idea is that if you can dig through that great 9-mile divide quickly and cheaply enough, the great problem is settled. But the engineers on the Isthmus have learned that that is not the whole problem. They have the equipment there, so that, given money and time enough, they could make a sea-level canal through the divide. That part would be simple digging. But when you come to the 22-mile section in the lower Chagres valley, difficulties begin to arise. It is swamp up to Bohio, and above that point there are large areas of swamp. Near Gatun the ground is, perhaps, 5 or 10 ft. above sea-level, and it rises continuously to about 25 ft. at Bohio and 50 ft. at Gamboa. Now, if you dig a sea-level canal through that stretch, you would have a narrow, winding ditch, with its bottom anywhere from 50 to 90 ft. or more below the level of the adjacent country. We do not know what the material is in all of that section, but a great deal of it is soft swamp. What slope is that material going to take? The

Panama railroad crosses a place called the "Black Swamp," where they drove piles 100 ft. and did not find bottom, and 500 ft. of track sank in one night. With a rainfall averaging 20 in. a month during a large part of the rainy season, the flat slopes on the sides of this ditch would be continually washing into the canal.

With regard to the rivers: There are 15 or 20 tributary streams, of considerable size in the season of floods, entering the Chagres between Gamboa and Gatun. It is a very difficult matter to control those streams. If taken into the canal, they would be all the time carrying the swamp and its material into the ditch. You might build dams, but there is nothing to put dams on. Practically, what you would have to do if you built a sea-level canal would be to build three canals. You would have to make a sea-level ditch and diversion-channels on each side to take care of the tributary streams.

Tracings from the latest official maps show comparative sailing-courses of the sea-level and lock-canals. The lock-canal gives free sailing in straight courses. The channel is 1,000 ft. wide most of the way through the Gatun lake. In contrast with this, the proposed sea-level canal is winding as well as narrow. Its curves are necessary to avoid the hills. These curves look very well on paper, but would be difficult of navigation. Recently the lock-canal channel, which was originally proposed to be 200 ft. through the Culebra divide, has been widened to 300 ft. That widening can be done, because the material taken out is so much less with the lock-canal than with the sea-level canal that the widening is not an enormously expensive task.

Captain Oakes, like many other engineers, is a little worried about the Gatun dam. My opinion is that it will be the safest dam that was ever built. Its size is preposterous compared with the stress upon it. As far as slides from its slopes are concerned, there is no danger. They are putting in rocks there the size of an arm-chair. A rock-fill of that sort does not easily slide. Then there is a common idea that the dam is porous, and that water will percolate through it like a sieve. I was interested to find that the material is not sand, but clay. Even the material that is being dumped as waste makes the finest kind of a dam. When the original borings were made

at the Gatun dam site, while the International Board was studying the subject, they penetrated two deep geological gorges, filled with clay. In those gorges, about 200 ft. under the ground-surface, the wash-drills brought up some stones about the size of wheat-grains, and it was duly reported that there was a stratum of gravel down there. Later on, they put down diamond-drills, and brought up solid cores from that supposed gravel, and found these gravel-stones were imbedded in solid impervious clay. They gave me a sample when I was down there of that supposed gravel. I have it with me to-day, and I will ask you to estimate how much water you think will go through it.

The other argument against the lock-canal is that there will be difficulty in passing through locks. But every vessel which passes through the Manchester ship-canal has to pass through four locks. The traffic through the Sault canal, the largest in the world, has to pass through a great lock, and in all the years that those two canals have been open there has never been a serious accident in locking a vessel. During that same time the narrow channels connecting the Great Lakes have been repeatedly obstructed by vessels going aground. You can see, of course, that those channels are a close parallel to a Panama sea-level canal. I have gone carefully over the designs for those great locks and I can say that there never were locks so carefully planned, in which the advantages of modern machinery have been so fully utilized, or in which contingencies affecting the safety of the vessels have been so carefully considered and estimated. The vessel will not be moved through the locks under her own steam. She will come up to the approach pier and there will be taken in charge of electric towing-machinery and that will move her through those locks in absolute position, keeping her off the lock-walls. It is possible in this way to handle a vessel through those locks with greater safety than in approaching her own pier after a voyage.

The men who are doing the work on the Isthmus have given careful, unprejudiced study to all of these questions. I was informed that some of them went there with their minds rather impressed with the sea-level idea; but they are building to-day not merely the cheapest canal—though it is the cheapest—but the one which will be the safest and most convenient for the world's ocean traffic.

JOHN D. EVANS, Lowell, Mass.:—The dangers attending the construction of a lock-canal at Panama have been set forth in this paper and elsewhere by Mr. Granger and by many others. I understand the purpose of the present discussion in this place to be, not to criticise or condemn the methods of work now in progress, but to reconsider the whole question of the best practical type of canal in the light of the latest information. Mr. Granger has re-stated the case with some additional facts and arguments bearing upon various elements of the problem, particularly the money and time required for the completion of a sea-level canal. Without going over all the points involved, I may briefly add my observation and experience as to one or two of them.

As to the size of the locks required for that type of canal, it seems reasonable to expect that even the size now proposed will be found hereafter to be so small as to limit the progress of shipbuilding in that respect. The *Great Eastern*, built not very long ago, was 680 ft. long. It was a practical failure, but not by reason of its length or structural design; and we now have ships in use nearly 800 ft. long and larger ones under construction.

In 1900 I was an assistant engineer for the Isthmian Canal Commission, and I remember that we first made our surveys and borings for locks 800 ft. long. Before this work was completed we received instructions to extend them, so as to provide for 1,000-ft. locks; and for this size we subsequently made our plans at Washington.

In 1901–1902, before our government took up the work of construction, I walked over the more important parts of the line to satisfy my own curiosity, inspecting particularly the Culebra cut. Whether Mr. Granger's proposed use of flumes, etc., should be adopted or not, I see no reason why night-work could not be economically carried on, with the result of reducing the time required for a sea-level canal to a shorter period than will certainly be necessary for the completion of locks and dams, as well as excavations for a lock-canal. Upon the basis of the estimate of 37,000,000 cu. yd. per year under the present system of a single 8-hr. shift, the excavation of 93,000,000 cu. yd. by 24 hr. of daily work seems reasonable. This would finish the sea-level canal before 1915.

With regard to the occurrence of earthquakes, and the amount of possible damage from that source, I would say that while at work in 1900 on a survey of the western division of the Nicaragua canal, I felt the shock and observed the effects of an earthquake serious enough to damage masonry structures. On the coast of the Pacific, 8 miles away, it produced violent undulations in the sandy beach, and was followed by a tidal wave of unusual force, which penetrated far inland. I believe it would have beached any masonry structure in the line of its movement.

In my judgment, the sea-level type is the proper one for the Panama canal.

ERNEST HOWE, Newport, R. I.:—Two objections to a lock-canal raised by Mr. Granger in his plea for a sea-level canal at Panama—the possibility of earthquakes, and the character of the foundations for the dam at Gatun—do not seem to me to have the weight which others, as well as he, have given to them.

Mr. Granger, in discussing "The Uncertainty of Foundations," after referring to the insufficient grounds that the French had for believing that the foundations at Gatun were satisfactory, says that the American engineers in charge of the work have made numerous additional borings, and successive reports from them have indicated unmistakably the continuance of more or less doubt as to this important element. This, and other remarks that follow, while not actually asserting the impossibility of finding safe foundations, at least convey by implication the impression that the dams and locks are to rest upon unsafe materials.

I have described elsewhere [10] the character of the rocks at the different lock-sites and the alluvium that fills the Pleistocene valleys at the Gatun dam-site, and C. M. Saville [11] has made a long series of experiments to test the suitability of the materials used in the construction of the dam. It is unnecessary to re-

[10] *Annual Report of the Isthmian Canal Commission*, 1907, Appendix E; *Report on the Geology of the Canal Zone*, pp. 124 to 130. Isthmian Geology and the Panama Canal, *Economic Geology*, vol. ii., No. 7, pp. 639 to 658 (Oct.–Nov., 1907).

[11] *Annual Report of the Isthmian Canal Commission*, 1908, Appendix E; *Report of C. M. Saville, Assistant Engineer, on Gatum Dam investigations*, pp. 127–196.

peat this evidence; but I may say that careful study of many borings, and familiarity with the surface geology, have led me to conclude that the locks will rest upon solid rock, and that the dam, built of the same materials as its much-abused foundations, will be homogeneous from bed-rock, 300 ft. below sea-level, to crest. I do not believe that any one, fully conversant with the conditions at Gatun, can doubt for a moment the safety of the foundations for locks or dam.

Earthquakes are not unknown at Panama; and yet I do not think it rash to say that the likelihood of destructive shocks is remote. A study of the causes of Central American earthquakes shows that a majority of the shocks are closely associated with active volcanism. There are no active volcanoes nearer the Canal Zone than Costa Rica. Tectonic shocks originate along lines of weakness in the earth's crust; and to this class belong the earthquakes of the Greater Antilles, Colombia and Venezuela. According to Bertrand,[12] a study of these lines of weakness and the areas of volcanic and seismic activity associated with them shows that Panama is situated in "a sort of dead angle," and lies equidistant, N–S., from the nearest points subject to disturbance. Be this as it may, it has always seemed to me that any sea-level canal smaller than the "Straits of Panama" (proposed by some, without regard to cost) would prove as vulnerable to earthquakes as a lock-canal, because any shock violent enough to destroy the locks would unquestionably cause land-slides of such magnitude in the region between Bas Obispo and Pedro Miguel as would effectually close the canal to navigation for a period quite as long as would be necessary to repair the damage to the locks.

I would add that I am no longer in any way connected with the Canal Commission, and that my conclusions as to the geology of the Isthmus are based upon personal observation, and have not been influenced in any way by the opinions or wishes of others.

Henry G. Granger, New York, N. Y.:—With regard to Mr. Baker's remarks about the swamp in the region of Gatun

[12] Bertrand, Marcel, "Les Phénomènes Volcanique et les Tremblements de Terre de l'Amérique Centrale." *Rapport de la Commission. Compagnie Nouvelle du Canal de Panamá*, Annexe I, pp. 121 to 134, Paris, 1899.

—that the sides would keep sliding in constantly, and that it would need a whole fleet of dredges—I would cite the history of the Suez canal, and call attention to the fact that the lower Atrato river runs through many miles of swamp, to which the land along this canal is not to be compared. Right at the edge of the river, within 6 ft. from the bank of this bottomless swamp, you can let down a plumb-line for 48 ft. and not find bottom; and that is not in one spot, but all over that section of the swamp. I do not think the swamp feature important, especially when we have a flow of tide in and out that would carry such deposit as would run in. A single dredge would probably take care of any deposit in the mouth of the canal.

Referring to the paper by Lewis M. Haupt, former member of the Nicaragua and Isthmian Canal Commissions, I am pleased to see that, although one of the most enthusiastic advocates of the Nicaragua route, he now clearly declares that this is no longer an issue, and that the government is bound to make good at Panama. With his statement that the vital risk is seismic convulsion, I am in thorough accord; but from my extended experience in lofty mountain ranges, with very steep slopes, and subject to earthquakes, I am convinced that when a given class of material has reached its natural angle of repose, it makes no difference what the vertical distance may be between the planes of the upper and lower termini of the slope. An earthquake that causes a severe land-slide is a most unusual occurrence. To cause a true slide, it would appear that the oscillation would have to be such that at the maximum swing the material would be outside of its natural angle of repose. I am strongly inclined to believe that the few reported slides (all unimportant) were due to the dropping-off of weathered material which no longer had the nature adapting it to the angle of repose at which it was found when the shake occurred. I cannot believe that, in a sea-level canal at Panama, of a minimum bottom-width of 300 ft., the material on the sides being excavated to its natural angle of repose in the first place, there would be any possibility of a slide that would obstruct traffic.

In regard to Mr. Haupt's final paragraph, I may cite the following extracts from a letter addressed to me, Feb. 9, 1909, by C. von Philp, manager of the machinery department of the Bethlehem Steel Co., in reply to my inquiry whether that com-

pany would undertake, for a price to be fixed, to design in detail, build, and guarantee a machine such as I had proposed in my paper:

". . . I have made a study of your plans and description, both of the general impact process, on which your machine is based, and with special reference to the machine as you desire to build it, in combination with shovels, belts, and traveling carrier for loading a train of cars, and all of sufficient strength to stand the strains of a rapidly reciprocating ram of from 10 to 13 tons' weight.

"The recent history of submarine gravity-ram system rock-excavation establishes the fact that a given average amount of rock can be dislodged or shattered by a blow of a given force. The quantity in each case varying with the class of rock and the force of the blow.

"Your problem is to make your machine so that the ram reciprocates in a cylinder and guide, suspended in such a way that the blows for tunnel-work may be readily directed at any portion of the tunnel-face, and that the ram should not necessarily stop working in order to change the direction of the blows. Further, the suspension must be in such a manner that glancing blows in any direction will not cause injury to the machine. The shovels must be so placed as to cover the entire floor of the tunnel-heading with their combined radii, and so arranged as not to interfere with the ram working at any position. Outside of these points, your proposition is one of simple every-day engineering principles, applied to the particular purpose you have in view.

"The recoil-strains in our gun manufacture are much more serious than anything that will be encountered in your operation, either for regular work or for missed shots; and I can foresee no serious difficulties in making this machine with the features of varying the length and rapidity of stroke you desire, in order to excavate several feet of tunnel before moving the entire machine forward. There may be some details that it will be advisable to modify later on, and for future machines; but I anticipate that none of them will be of any serious nature.

"I can see nothing in this machine that cannot be made to perform its functions as intended. Any particular rate of progress in tunneling, of course, we could not guarantee, as this would vary both with the character of the rock and the skill of the operators.

"With 800 h.p. applied to the air-compressors, there should be no trouble to strike 60 to 100 blows per minute of the desired force.

"I will say, in this connection, that the men, as a rule, will get more and more experienced in the handling of the machine, and will strike as many blows as they possibly can in order to get out the maximum quantity of rock. This refers to men employed on piece-work, which is quite natural in work of this nature.

"We will design this machine for you under my personal supervision, and manufacture the same after the design has been approved by you, and deliver the machine to you after a trial-run in our yard or in this vicinity, to show that the various parts are properly designed and constructed to perform their functions as intended.

"We will guarantee its proper working with a minimum breakage, by which I mean such breakage as will develop in machines of this class, such as steam-hammers, presses, dredges, etc. But the breakage to be only such as would occur through no fault of material or design. Under this guarantee we will supply sufficient wearing parts for the first six months' operation, and replace any breakages that occur during this period. . . ."

The concluding portion of the letter, here omitted, concerns other conditions of the contract, such as patent-rights for improvements, etc., but does not affect the statements and the offer of guaranty quoted. It will be noticed that this guaranty is for 60 to 100 blows per minute, instead of 40, as estimated by me.

As to Mr. Manton's suggestion concerning a subsequent excavation to sea-level, without serious interference with traffic, I would say that I do not believe it economically practicable to excavate to sea-level from a previously excavated high level, as this would involve working, both for rock-shattering and for dredging, through a depth from plus 60 ft. to minus 40 ft., or 100 ft. While it would be within the bounds of possibility to make locks so that they could be broken away as water should be lowered by successive steps, I do not regard such a construction as certain to be practicable and satisfactory, even aside from the doubtful features as to the stability of the dam and the reservoir's retention of water. Especially do I regard any such system as unadvisable, when it is demonstrable that the canal can be put through at sea-level, with a minimum bottom-width of 300 ft., in as little time as would be required to install the locks and dams.

With regard to Mr. Manton's pertinent remark that, in view of the present admirable progress, it would seem a mistake to abandon the present program, I will say that on consideration of the recent equipment-purchases and organization, I have come to a similar conclusion. And although I am convinced that the particular method of fluming-out the material suggested in my paper would be the most rapid if used from the inception of the enterprise, I frankly admit that, under the present circumstances, with the equipment and organization as they are, I believe it advisable on economic grounds to continue the work as nearly as practicable by the methods already in use.

With regard to the erosion in the flume-lining, to which Mr. Manton calls attention, I would say that, according to my experience, the erosion of chute-plates in dredging is not caused by the material carried in water, which exercises but a very small fraction of the abrading-power of rock or sand dry, or simply damp material, as is proven by the great quan-

tities of gravel, sand, and rock pumped through the relatively thin pipes leading from hydraulic dredges to the distribution-ground.

In view of my tunneling-machine not being a rotary chipper, completely filling the heading, I do not anticipate the troubles that have been found in such machines owing to lack of clearance, etc.

As to a comparison between the results of my machine in mining- and tunneling-work and the Lobnitz in submarine work, I would refer to the publication of the Engineering Conference of 1907, where it is shown by Mr. Hunter (a sea-level member of the Panama Board) that the effect of ram-shattering is a series of inverted cones that are carried far below the line to which it is desired to break. Beverly R. Value lately told me that in the work of the Empire Engineering Corporation, in the extremely hard flint at Buffalo, it was found that, after dredging off a given amount of rock shattered, the second penetration of the ram was much easier, owing to the cracks and shattering having extended far below the line to which it was possible to dredge. Mr. Value prophesied that in tunnel-work my machine would require fewer blows per yard, by reason of the material falling away and presenting a clean face, than in the Lobnitz method, where the material remains in place and acts as a cushion to the blow. When William Barclay Parsons gave me the suggestions that I acknowledged in my paper, he added: "If you get your machine strong enough to stand up against the vibration and racking that it will be subject to, you have got it."

I appreciate Mr. Manton's remarks as to the tunnel-shovel, and quite agree with him that it will be necessary to have the teeth set in such a way that the bottom can be skimmed without constantly catching on the serrated floor.

I am indebted to Captain Oakes for his extended criticism of my paper. Some of his remarks as to earthquakes have been covered in my reply to Professor Haupt. I think his statement that an earthquake severe enough to rupture the Gatun dam and allow the water to destroy Colon and the intervening valley would probably cause the destruction of the city of Colon without any flood, is unwarranted. To destroy the locks and all imaginable safety-devices, and let the water

out of the lake, it is simply necessary that the movement be sufficient to jam the gates, or doors, and bridges—a movement not necessarily destructive to the edifices of Colon, although likely enough to be felt there. It is conceded that, if an earthquake should open a slight fissure in the dam or in its foundations, the entire body of the lake would cut itself out. The loss of water, either through mishap in the locks, or by the cutting out of the dam, would imply the loss of many months, irrespective of repairs, before the lake would re-fill sufficiently to permit the resumption of traffic.

In my judgment, the record of tremors, light shocks, shocks sufficient to drive people out of their houses, and shocks of destructive force, on the Isthmus of Panama, is sufficient to condemn as unwise and unsafe any engineering-works in that locality which would be vulnerable to seismic action. General Abbot's book, quoted in my paper, is clear on this subject; and other available records warrant my conclusion.

[I here present extracts from a book entitled *Five Years at Panama*, written by Wolfred Nelson, M.D., and published by ——— ——— in 190–. The statements contained in these extracts have been corroborated by trustworthy persons residing on the Isthmus. They describe an earthquake which occurred at Panama in September, 1882, and which shook many ordinary buildings violently; threw down the upper part of the façade of the cathedral; wrecked the town-hall, breaking and precipitating into the Plaza many columns and arches, destroying the front entirely and causing part of the roof to fall; badly cracked the office-building of the Canal Co.; and shook down the old tower of the "Chapel of Ease." On the island of Toboga, nine miles from Panama, part of a solid cliff fell into the sea. At Colon the shock was still more violent. For twenty-two miles between Colon and Baila-Mona the Panama railroad was made almost useless. It sunk in some places, and was thrown completely out of line in others. The 600-ft. bridge at Barbaccas was thrown slightly (according to another authority, 1.5 ft.) out of line. At Crux (now called Las Cruces), near one of the central stations on the railroad, a substantial stone church was shaken to pieces, leaving no fragment of wall as high as 4 ft. After describing the minor shocks of the following five days, Dr. Nelson says he afterwards learned by inquiry that the earthquake which, in the autumn of 1858, so greatly damaged Cartagena, on the Atlantic coast, had done much injury in the city of Panama also, and, moreover, that "upwards of a century ago" the Panama region had been terribly shaken.]

As to the military considerations, which Captain Oakes treats so lightly, I do not deny that a sea-level canal could be obstructed by the sinking of a vessel in the channel; but such an obstruction, if attacked by a competent wrecking-corps, could not close the canal for more than two weeks. Captain

Oakes admits that a secret agent, operating alone, or with other persons, could destroy the lock-walls or the lock-gates. Could this injury be repaired in two weeks?

With regard to the danger which Captain Oakes declares to be beyond his imagination, that sufficient nitro-glycerine or other explosives could be carried over a lock-site and dropped from an air-ship with sufficient accuracy to do material damage, I call attention to recent aëroplane tests repeatedly made by the Wright brothers, which are reported to prove the practicability of this very performance. Rear-Admiral Robley D. Evans[13] said:

> "And let us remember always what delicate things the lock-gates of a canal are, and how powerful the ram of a battleship is when moved even very slowly, with 26,000 tons behind it. And above all, let us not forget what one stick of dynamite could do to such lock-gates; and there are always brave, daring men in every war who will freely give their lives to injure an enemy."

In view of the elaborate precautions mentioned by Captain Oakes against accidents to the locks, I ask whether it is not conceivable that an accident might take place in such a manner as to destroy both safeguards and locks. He does not deny the possibility of accidents, but claims that they would be infrequent. One would be enough. I agree with him that accidents from fogs and winds will be rare; but I reiterate my opinion that they will be more probable and more serious in a lock-canal. Shipping is occasionally compelled to leave the port of Colon on account of the severe wind-storms.

I agree with Captain Oakes in doubting the existence of serious danger to an earthen dam from tropical burrowers; but it is not shown that there is no danger from this source; and as to his assertion that this would be an equal menace to dams involved by the sea-level canal, I would say that my proposition involves the building of no dam to hold in check a body of water that would be the slightest menace.

The widening of the canal to a minimum of 300 ft. not only improves its navigability but obviates all difficulty from the Chagres, except for a few hours once in many months, in times of most severe flood.

As to the water-supply for a lock-canal, General Abbot, in his exhaustive work,[14] was my authority for the statement that

[13] New York *American*, Feb. 11, 1906. [14] *Problems of the Panama Canal* (1905).

by reason of lack of supply alone there would be, in extreme droughts, insufficient water for lock-purposes with an active traffic. There is no certainty that the estimated allowances for evaporation, leakage, filtration, etc., are correct, because exact data are lacking. Some of us believe it quite possible that the Gatun lake might not hold water enough to pass a row-boat through the locks.

Captain Oakes frankly says: "If a sea-level canal of equal capacity and facility of traversing could be constructed for the same cost as the lock-canal, or less, there would be no room for argument." I may fairly add that in estimating relative costs, the lock-canal should be charged with an amount representing the capitalized extra expense of operating it. If a sea-level canal cost practically nothing to operate, and a lock-canal cost, for the sake of argument, $4,000,000 per year, at 2 per cent. (the interest of the canal bonds), the sea-level canal might be $200,000,000 more expensive in actual first cost, and still be as economical and produce the same net revenue as the lock-canal.

In the extracts from the reports of Major Sibert, Major Harding, and Mr. Saville, quoted by Captain Oakes, it is noteworthy that not one of these three reports is absolutely positive. Major Sibert introduces an "It is thought;" Major Harding, an "if;" and Mr. Saville, another "if." Against opinions thus qualified may be placed the outspoken judgment of numerous competent engineers who consider the construction of the proposed Gatun dam and locks on such foundations unjustifiable.

With regard to the complete change of equipment and method involved in the adoption of the flume-system, suggested in my paper, I need only refer to what I have already said in my reply to Mr. Manton, and add that in my letter of Jan. 13, 1909 (21 days before the date of Captain Oakes's communication), to the New York *Tribune*, I expressed my belief that, under present methods, sufficient speed of construction could be maintained to produce a sea-level canal in the same time as the lock-canal; and that, now that the overhead charges for sanitation and preparation are being so materially reduced, both in aggregate amount and per cubic yard, the cost of the work as at present conducted is getting down pretty close to the figures of Mr. Wallace's predictions. Therefore, I

unhesitatingly advocate the continuance of the present excavation to its maximum capacity.

I am not advocating any plan or device whatever. I simply advocate the construction of the Panama canal at sea-level, and cheerfully stand for whatever means will produce the result of inter-oceanic navigation at sea-level in the least time. The machines and devices I suggested were mentioned because, if not separately described, my right to my inventions might be jeoparded; but I freely offered the government the right to utilize any or all of them in the construction of a sea-level canal by making its own terms with the manufacturers. As to the criticisms of Captain Oakes on these machines, I beg to offer the following observations:

I admit that, with the existing equipment and conditions at Panama, it would probably be inadvisable to install my grading and pile-driving machine. This machine consists of a locomotive-crane and boom, and is intended for construction-work at the end of an isolated, inaccessible railroad-track. Either leads, shovel, or bucket could readily be substituted for it. It was designed for specific conditions in a specific piece of work, and is of evident efficiency for its purpose.

The suction-dredge I proposed is described and illustrated in my paper, Hydraulic Dredging for Gold-Bearing Gravels.[15] It was designed, and I expect will shortly be built, for harbor- and dock-work, where the same crew and power-plant can at will work in matted roots and stumps on the surface, or pump material, free of obstruction, from below, and, if desired, do both at the same time, with a capacity depending upon sufficient power only.

The electric transmission of power generated by producer-gas was suggested, because, in my opinion, for a completely new installation, it would, in my judgment, give the ideal economy both in horse-power developed for coal consumed and in freedom from trouble on the dredges, etc. Whoever has seen the fleet of electric gold-dredges at Oroville, Cal., cannot fail to appreciate that electricity offers the best distribution of power for such a fleet.

I did not give further details of my belt-method for making

[15] *Bulletin* No. 28, April, 1909, pp. 389 to 410.

fills, because I thought that the illustration was sufficient to make it evident. Captain Oakes does not understand how 10,000,000 cu. yd. of material could be deposited by such a machine without resetting it, or could be handled at any such rate as was stated. I beg to reply that my statement was not based on guess-work, and that I could furnish him with detailed drawings and estimates, together with guaranteed offers for the construction of the required plant. But I should add that I would consider it advisable not to use belts wider than 84 in., which would materially reduce the capacity of the apparatus, although still giving several times as much as Captain Oakes seems prepared to believe. I agree with Captain Oakes's criticism that my design "whereby a full train of cars could be loaded in a cut or tunnel behind one or two steam-shovels" could not be applied under the circumstances he assumes. Where trains can be run past a shovel, this device could not come into competition; for it certainly complicates the ordinary method of depositing directly into cars from the shovel. But it is designed only for tunnel-work, subway-work, or other peculiar situations where cars cannot be run past the shovel, and where switching would cause a greater loss of time and money than the cost and maintenance of such an equipment would involve.

As to the movable cantilever, Captain Oakes seems not to have noticed that it was specified for rock or gravel. In earthwork, it would unquestionably be unadvisable to use such an apparatus for more than one car, and possibly not of the largest size at that. I must confess, moreover, that the illustration of this cantilever, given in my paper, was a hasty drawing, made in a few hours, to illustrate the principle of its construction. Yet I am inclined to think that, with the balancing-weight and truss properly proportioned, his assumption of a weight of 400 tons on the end of the track is somewhat too high.

As to the cross-section diagram, I must frankly admit the justice of his criticism. The additional section blocked out in this diagram is, as he points out, one-fourth of the total and only one-third of the original. I regret the error, committed in the haste of my preparation of text and illustrations for early publication, and I thank Captain Oakes for correcting it. Yet I do not see that it affects essentially the force of my argument.

With regard to my statement of the yardage for a 200-ft. sea-level canal, quoted from the report of the Board of Consulting Engineers, which Captain Oakes says did not refer to a canal of that bottom-width, I think that my interpretation of the report was reasonable. But the amount of yardage is the important point; and for the purpose of this discussion, although not in accordance with my ideas from other sources as to the yardage of a canal 300 ft. wide, I will accept the mean between his two figures of 400,000,000 and 500,000,000, or, say, 450,000,000 cubic yards.

Captain Oakes has correctly comprehended my proposition regarding the Chagres. In reply to his question as to navigation on the Magdalena and Atrato rivers, I would say that the maximum tonnage of boats on these streams is about 300 tons. Some of them are sea-going. My experience is that flat-bottom boats are much less manageable than deep-draft boats. He is mistaken in saying that "no master would risk his vessel in a 300-ft. channel with a current of 5 miles per hour." Any competent captain would unhesitatingly take his vessel against a current of this velocity, although, to go with the current in this width, unless on absolute straight-of-way, he would wait until the tide had gone down to 2.5 miles per hour or thereabouts. I say this advisedly, after consultation with experienced deep-sea navigators.

A tidal current might be very beneficial by floating, after a few hours' delay, a vessel accidentally grounded at any time except extreme high tide. But a vessel grounded in the proposed lake would be compelled to undergo a very long delay and the unloading of a part of its cargo.

Being familiar with the rapid and excessive floods of tropical streams, and their effect on their bottoms and banks, I anticipate no erosions by Chagres floods, the deposits of which could not be readily handled by a single dredge.

My estimates as to the cost of jetties were based on the statement by one of the high officials on the Isthmus, that the jetties could be made of material selected from the cut, and on the consequent assumption that the cost chargeable to the jetty-construction would be only that of the long haul and dumping of such material. I see no reason for the huge stones that Captain Oakes suggests as covering, in view of the proposition

to dump material on the outside of these jetties, and thus form shoals, which would protect them. Of course, my idea in using the cantilever in jetty-building was to run it out on track laid as the jetty was extended. I see no need of a trestle to carry the cantilever, as Captain Oakes suggests. The main object of the cantilever is to do away with trestle-work. The rock- or gravel-fill should not push out at the end any more than from the sides, provided it had reached its natural slope. If the "immense" weight of this cantilever could not be handled on a rock-fill, with its weight distributed at the rate of 2 tons per square foot at the end of the outer 12 or 14 ft. of the proposed cantilever, then there is not a safe railroad-embankment in the United States to-day, for this load per square foot is exceeded constantly on all roads. Rock freshly dumped may settle a little, but will not flow.

No longer being an advocate of the flume-system for the present conditions on the Isthmus, I will simply say as to this point of Captain Oakes's criticism that I have seen sluices in different fields, and have had them blocked repeatedly, the water being stopped with the material in the sluice, and that any miner or hydraulic-fill man can readily regulate his water so as to avoid any trouble on account of a possible shut-down.

My estimate for pumping, which Captain Oakes has overlooked, will be found on p. 25 of my paper.

If the flume had been constructed, the Chagres would be allowed to flow in its present course until the apron was completed, and then a cut made allowing it to go over the apron, and the excess over the flume-supply would then resume its way to the Atlantic until the canal was cut through. The unit-prices mentioned in my paper are, in view of the fact that they are simply for labor and repairs, very much above current practice, and give a large margin of safety.

Up to date practically all of the work that has been done is work that would be equally necessary under the sea-level type, and but a very small percentage (perhaps less than 1 per cent.) of what has been done towards the lock- and dam-construction would be wasted if a change were made now. That a change will have to be made eventually through the failure of the lock plan is not merely my own opinion, but that of many other engineers.

Since writing the foregoing, I have received from Senator Kittredge the full text of the report of the Board of Engineers who recently accompanied Mr. Taft to the Isthmus. As this report cover points exclusively brought out in my paper, I beg to offer the following comment:

Type of Canal.

Page 8. "In view of the fact . . . that the excavation . . . is being made somewhat more rapidly than was anticipated, we have considered in a very general way the relative cost and time of construction of a sea-level canal."

The estimate was 1,000,000 cu. yd. per month; the results are over 3,000,000 cu. yd. per month. Is their reference to this great administrative achievement of Colonel Goethals and his able assistants as being "somewhat more" either a fair, reasonable, or true statement of the situation, and does it evidence fair-mindedness, or an unswerving determination against all consideration of the overwhelming arguments in favor ot sea-level?

Page 9. Here is mentioned as necessary the construction of a Chagres regulating-dam at Gamboa as an essential preliminary to excavation of a sea-level canal. My apron proposition, in connection with a sea-level canal of 300 ft. minimum width, does away with all question of a menacing reservoir on the Chagres. This point was apparently not considered by them, although before the Secretary of the Institute consented to publish my paper it was favorably passed on by an engineer of the highest standing and position.

Rock-Excavation Under Water.

Pages 9 and 10. "Much has been said about the economy of excavating rock under water by modern appliances as compared with the cost of such excavation in the dry with steam-shovels after blasting.

"We concur in the opinion of those in charge of the work at the Isthmus that it is more economical, where the conditions are favorable, to excavate rock in the dry than by any under-water process now in use. Experience is not yet available to us which will justify the belief that, with the depth of cut and the quality of rock found on the Isthmus, the general adoption of the subaqueous methods would prove more expeditious or cheaper.

"It is probable that more economical subaqueous methods will be sometime developed, but it would not be wise to base a change in plan of important work upon prospective results to be obtained by any method not yet thoroughly tried."

This Board of distinguished engineers evidently did not comprehend that the impact rock-shattering ram could be

attached to a steam-hammer, properly suspended on a barge, in one or multiple units, and thus strike blows of much greater force and many times the rapidity of the gravity system. The records are available of the marvelously economical execution of the gravity-ram in submarine rock-excavation of all classes since the time of Mr. Wallace's early work on the Mississippi, including its extended use in many countries, and its high indorsement by the Chief Engineers of the Suez canal and of the Manchester ship-canal, where it was used with such highly satisfactory results; and now it is rapidly and economically excavating the hardest flint on the New York barge-canal.

As to earthquake hazards, the report says:

Page 10. "It has been suggested that the canal region is liable to earthquake shocks, and that a sea-level canal would be less subject to injury by earthquakes than a lock-canal.

"We have seen, in the City of Panama, the ruins of an old church, said to have been destroyed by fire, containing a long and extremely flat arch of great age, which convinces us that there has been no earthquake shock on the Isthmus during the one hundred and fifty years, more or less, that this structure has been in existence, that would have injured the work proposed."

Since the time of Pliny, in earthquake countries the shelter of the masonry arch has always been regarded as the safest place in time of a shock. I am surprised that they did not recall, before making this statement, the photographs that have appeared in the public press of the past few weeks, showing many arches intact in the walls left standing at Messina and Reggio; and I wonder that engineers of unquestioned high standing should refer to an arch in connection with earthquakes, not realizing the mathematical necessity that, for an arch to fall, even provided that, as so frequently is the case in ancient Spanish architecture, the cement be not stronger than the stones it unites, it is necessary for the side pillars to separate a distance sufficient for the versed-sine of the arch to settle first to zero. Thus it is mathematically necessary, in order for an arch to fall, that the side supports should separate a distance equal to double the difference of half the chord of the arch and the chord of half the arch.

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Bulletin of the American Institute of Mining Engineers.

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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.

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SECTION I.—INSTITUTE ANNOUNCEMENTS.

This section contains announcements of general interest to the members of the Institute, but not always of sufficient permanent value to warrant republication in the volumes of the *Transactions*.

SECTION II.—TECHNICAL PAPERS AND DISCUSSIONS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

A detailed list of the papers contained in this section is given in the Table of Contents. They have been so printed and arranged (blank pages being left when necessary) that they can be separately removed for classified filing, or other independent use.

A small stock of separate pamphlets, duplicating the technical papers given in Section II. of this Bulletin, is reserved for those who desire extra copies of any single paper.

Comments or criticisms upon all papers given in this section, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

All communications concerning the contents of this Bulletin should be addressed to Dr. Joseph Struthers, Assistant Secretary and Editor, 29 W. 39th St., New York, N. Y. (Telephone number 4600 Bryant).

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

INSTITUTE ANNOUNCEMENTS.

Spokane Meeting and Excursions.

Further details of the 97th meeting of the Institute, at Spokane, have been sent to members in the Special Circular of May 8, 1909, and for convenience a summary of the additional information is given later in this announcement.

In spite of the present small number who have joined the party, final arrangements have been made to complete the itinerary given in the circular. It is greatly to be desired, both in behalf of the Institute and in recognition of the special courtesies to be extended on the trip, that the party be made larger. Exceptional facilities to visit the many mines and smelters have been freely provided; and members are requested to make all reasonable effort to attend the meeting at Spokane and participate in the excursions.

For those who cannot take the entire trip, it may be possible to provide special Pullman accommodations after leaving the Yellowstone Park. In order to make the necessary arrangements, however, the names of members and guests contemplating this partial trip should be sent at once to Mr. Theodore Dwight, 29 West 39th St., New York, N. Y.

A special train of Pullman cars will leave Chicago September 16. This train will be side-tracked at all places visited, and the sleepers will be available for lodging and the dining-car for meals during the entire trip, except while taking the tour through the Yellowstone Park.

The general cost of the trip per member will be $300, which includes transportation, berth, and meals for the entire trip from Chicago back to Chicago, almost 6,000 miles, and the tour through Yellowstone Park, occupying in all about 30 days. Special accommodations on the train will be furnished at the following rates:

Chicago to Chicago, including transportation, berth, and meals,	$300.00
State-room for one person, $200.00 extra, or	500.00
State-room for two persons, total,	700.00
Drawing-room for two persons, total,	800.00
Drawing-room for three, total,	1,100.00

The excursions connected with the Spokane Meeting will have much of professional interest: the varied formations in the Yellowstone Park; the great copper-plants at Butte and Anaconda; the mining and metallurgical enterprises at Cœur d'Alene; the mining exhibits at the Alaska-Yukon-Pacific Exposition, Seattle; the Tacoma Smelter at Tacoma; and the mining and metallurgical operations in the vicinity of Salt Lake, at Bingham, Murray, and Garfield; and (possibly) the steel-plant and lead-smelter at Pueblo.

The scenic interest can be appreciated by mentioning the Yellowstone Park; the Cœur d'Alene district; the daylight ride from Spokane to Seattle; the Cliff drive at Tacoma; the bold scenery of the Wasatch mountains, and the Royal Gorge. It would be very difficult to plan a scenic trip of greater variety and beauty.

The schedule of the Institute meeting was planned to conclude in time for members to attend the convention of the American Mining Congress, at Goldfield, originally set for October 15. A recent communication from the Secretary of the Congress gives a change of date to September 27, 1909—the exact time of the Institute sessions at Spokane. Owing to the lateness of the change of date by the Congress, it is impracticable to alter the date of the Institute sessions.

The list of those intending to go on the excursion is filling up, and since preference in position of state-room or berth is given in the order of application, members and guests who expect to accompany the party and have not yet communicated with Mr. Dwight, are earnestly requested to do so without delay. There is still room for more.

Meetings of Other Societies.

The American Mining Congress.—The American Mining Congress will hold its twelfth annual session at Goldfield, Nev., Sept. 27 to Oct. 2, 1909.

During the past year several Committees have been investigating certain problems affecting the mining industry, and their reports will be presented and discussed at this meeting, as follows:

Vertical Side Line Law; by George W. Riter, Salt Lake, Utah, *Chairman.*
Coal Tax Insurance Fund; by Samuel A. Taylor, Pittsburg, Pa., *Chairman.*
General Revision of Mining Laws; by Walter R. Ingalls, New York, N. Y., *Chairman.*
Standardization of Electrical Equipment; by Dr. Edward B. Rosa, Washington, D. C., *Chairman.*
Prevention of Mine Accidents; by Dr. H. Foster Bain, San Francisco, Cal., *Chairman.*
National Forest Service; by Col. A. G. Brownlee, Denver, Colo., *Chairman.*
Revision of Alaskan Mining Laws; by J. L. Steele, Landlock, Alaska, *Chairman.*

In addition to these reports, two important papers by eminent authorities will be presented for discussion—one on More Efficient Mine Inspection, and the other on Means for Increasing the Use of Silver as Money.

The Mining Congress invites all members of the American Institute of Mining Engineers to be present, and extends to them the privilege of participating in the discussions.

Further information may be obtained by addressing the Secretary of the American Mining Congress, James F. Callbreath, Jr., 1510 Court Place, Denver, Colo.

International Congress for Mining, Metallurgy, Applied Mechanics, and Practical Geology, Düsseldorf, 1910.—At the closing session, July 1, 1905, of this Congress, held in connection with the Lüttich Exposition of that year, it was resolved to accept the invitation extended by the mining industries of Rheinland-Westphalia, and to hold the next Congress in that region.

In accordance with this resolution, the Congress will be called to meet in the latter part of June, 1910, at Düsseldorf. For this meeting, which will occupy about one week, extensive preparations are in progress, including visits to technical institutions and industrial establishments, excursions to geologically-interesting localities, etc., intended to illustrate the papers and addresses presented in the four sections above named.

The exact date, together with more detailed and definite particulars of the meeting, will be published later. Meanwhile, inquiries, suggestions, and announcements of proposed papers may be addressed to Dr. E. Schrödter, Chairman of the Committee of the Congress, Düsseldorf, Jacobistrasse 5, Prussia.

Office Facilities for Visiting Members.

A separate room in the suite occupied by the American Institute of Mining Engineers on the ninth floor of the United Engineering Society Building, has been equipped with furniture and telephone extension for the temporary use of members of the Institute or of sister societies, or visitors suitably accredited.

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In short, to those who own complete sets of the *Transactions*, this Index will be a great convenience; but to those who do not, it will be a professional necessity.

This volume is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. It has not been stereotyped, and the edition is limited to 1,600 copies. The price of the volume, bound in cloth, is $5, and bound in half-morocco to match the *Transactions*, $6. The delivery charges will be paid by the Institute on receipt of the above price.

Hydrographic Chart.

Owing to the great value to hydrographers of the chart contained in the paper, A Graphic Solution of Kutter's Formula, by L. I. Hewes and Joseph W. Roe (*Bulletin No.* 29, May, 1909, p. 454), a special edition for office or field use has been printed on durable cloth. Copies of this separate chart may be obtained, at a cost of 50 cents each, on application to the office of the Secretary of the Institute.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

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For the protection and convenience of members, the following rules have been adopted:

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Any person discovering a mutilation or defect in any book of the libraries is requested to report it to the librarian on duty.

Library Additions.

From June 15 to Aug. 1, 1909.

ALLOYS AND THEIR INDUSTRIAL APPLICATIONS. By E. F. Law. London, C. Griffin & Co., 1909. (Purchase.)

AMERICAN INSTITUTE OF ARCHITECTS. Annuary, 1909. Washington, 1909. (Gift.)

AMERICAN INSTITUTE OF MINING ENGINEERS. Transactions. Vol. 39. New York, 1909.

AMERICAN IRON AND STEEL ASSOCIATION. Annual Report, 1873–1907. Philadelphia, 1873–1908. (Gift of James M. Swank.)

AMERICAN IRON AND STEEL ASSOCIATION. Annual Statistical Report, 1909. Philadelphia, 1909. (Exchange.)

ANUARIO ESTADISTICO DE LA REPUBLICA ORIENTAL DEL URUGUAY. Tomo I., 1907–08. Montevideo, 1909. (Gift.)

APPLICATION OF THE SOUTH SHORE TRACTION COMPANY, for the Permission and Approval of this Commission for the Construction and Operation of an Extension of its Street Surface-Railroad from the Boundary Line Between Nassau and Queens Counties to the Queensboro Bridge Plaza, in the Borough of Queens, and for the Exercise of a Franchise to Operate Cars on the Queensboro Bridge. (Case No. 1032.) New York, 1909. (Gift of Public Service Commission for the First District, State of New York.)

BORN'S NEW PROCESS OF AMALGAMATION OF GOLD- AND SILVER-ORES, AND OTHER METALLIC MIXTURES. By R. E. Raspe. London, T. Cadell, 1791. (Gift of T. J. Hoover.)

BRIQUETTING TESTS AT THE UNITED STATES FUEL-TESTING PLANT, NORFOLK, VA., 1907–08. (Bulletin No. 385, U. S. Geological Survey.) By C. L. Wright. Washington, U. S. Government, 1909. (Exchange.)

BRITISH COLUMBIA—MINISTER OF MINES. Annual Report, 1908. Victoria, R. Wolfenden, 1909. (Exchange.)

BUCKS COUNTY HISTORICAL SOCIETY. Collection of Papers Read. Vols. 1–2. Easton, Chemical Publishing Co., 1909. (Gift of B. F. Fackenthal, Jr.)

[SECRETARY'S NOTE.—Bucks county, Pa., is in many ways one of the most interesting regions in the Atlantic States. To its picturesque scenery, varied geology, and abundant flora are added historical and prehistorical association of wide range, comprising the dwellings and work-shops of the Lenni Lenape and the Shawnees, before the advent of Europeans; the colonies of the Moravians and Mennonites; the scenes of Washington's greatest campaigns; almost the earliest American iron blast-furnace, etc. And the citizens of this county have done wisely in maintaining since 1880 a Historical Society, at the social gatherings of which a large number of papers have been read, preserving old traditions, or recording new discoveries, or narrating the lives of eminent leaders, or setting forth the reminiscences of the oldest inhabitants before they passed away. President Henry C. Mercer, in his preface to these volumes, says:

"The trance of William Tennant; the origin of Princeton University in the Log College; the coexistence of the mammoth with the North American Indian, as proved by the remarkable Indian carving found near Doylestown, known as the Lenape stone; the constant claim of the unfortunate John Fitch to the invention of the steamboat; the concealment of escaped slaves in Bucks county; the taking of lands from the Indians; the establishment of Christianity; the families, homes,

houses, customs and landmarks of the region—these and other subjects of wider or narrower importance formed the themes of many papers upon colonial history, the Revolutionary War, archæology, church history, folklore, and genealogy, which were contemporaneously printed in the scattered columns of the county newspapers. But they would have been finally forgotten or lost to the general public but for the liberality of Mr. B. F. Fackenthal, Jr., who has collected, corrected, arranged, and published the complete series in the following pages."

Mr. Fackenthal's liberality has been effectively reinforced by his industry, care, and skill. The work he undertook was worth doing; and it has been well done.—R. W. R.]

CANADA—MINES DEPARTMENT. Report on the Mining and Metallurgical Industries of Canada, 1907–1908. Ottawa, 1908. (Exchange.)

CARNEGIE TECHNICAL SCHOOL, PITTSBURG, PA. General Catalogue, 1909. (Exchange.)

CHEMISTRY AND LITERATURE OF BERYLLIUM. By C. L. Parsons. London, Williams & Norgate, 1909. (Purchase.)

CLAYWORKER'S HAND-BOOK. London, C. Griffin & Co., 1906. (Purchase.)

COAL. By J. Tonge. New York, D. Van Nostrand Co., 1907. (Purchase.)

COAL FIELDS OF SOUTHWESTERN PENNSYLVANIA, WASHINGTON, AND GREENE COUNTIES. By J. W. Boileau. N. p., 1907. (Purchase.)

COAL-SHIPMENT AND THE LAYING-OUT OF STAITHE HEADS, WITH SPECIAL REFERENCE TO ANTI-BREAKAGE APPLIANCES. By John Kirsopp. Newcastle-upon-Tyne, 1909. (Gift of Author.)

CONTRIBUTIONS TO ECONOMIC GEOLOGY, 1907. Pt. I.—Metals and Nonmetals, Except Fuels. (Bulletin No. 340, U. S. Geological Survey.) Washington, U. S. Government, 1909. (Exchange.)

CONTRIBUTIONS TO ECONOMIC GEOLOGY, 1907. Pt. II.—Coal and Lignite. (Bulletin No. 341, U. S. Geological Survey.) Washington, U. S. Government, 1909. (Exchange.)

COTTON PRODUCTION, 1908. (Bulletin No. 100, U. S. Census Bureau.) Washington, U. S. Government, 1909. (Exchange.)

ELECTRICAL INDUSTRIES OF PORTO RICO, 1907. (Bulletin No. 99, U. S. Census Bureau.) Washington, U. S. Government, 1908. (Exchange.)

ELECTRO-MAGNETIC ORE SEPARATION. By C. G. Gunther. New York, Hill Publishing Company, 1909. Price, $3 net. (Gift of Publishers.)

[SECRETARY'S NOTE.—According to Mr. Gunther's preface, this book has been prepared to gather into convenient form the published information on the magnetic separation of ores. It is a compilation, supplemented by the writer's own observation and correspondence, and including only material which he considers to have present commercial importance. Within this limited sphere, it will be found convenient and useful, although the student will wish that the author had gone a little further into the principles of the subject, or had furnished some hints as to the essential differences in the machines he describes. The only result at which he seems to have arrived is, that, "the process suitable to the treatment of the ore under consideration having been carefully chosen, it will be found that any one of several machines will perform the function of the actual separation." Mr. Gunther's proposition that magnetic separation in its own field is a useful adjunct to the specific-gravity processes, but not a competitor, except in the concentration of magnetic iron-ores, seems to sum up pretty fairly the results of practice so far.—R. W. R.]

ENGINEERING RECORD DIRECTORY OF MANUFACTURERS OF AND DEALERS IN ENGINEERS' AND CONTRACTORS' MACHINERY AND SUPPLIES. Ed. 2. New York, 1909. (Exchange.)

ENGINEERS' CLUB OF ST. LOUIS. Annual Bulletin, 14th. St. Louis, 1909. (Exchange.)

ENGINEERS' SOCIETY OF WESTERN PENNSYLVANIA. Charter, By-Laws, List of Members, 1909. Pittsburg, 1909. (Exchange.)

ESTADISTICA MINERA DE CHILE EN 1906 I 1907. Santiago de Chile, 1909. (Exchange.)

FAMILY TREE OF COAL-TAR. N. p., n. d. (Gift of Carbolineum Wood Preserving Co.)

FIRE RESISTIVE PROPERTIES OF VARIOUS BUILDING MATERIALS. (Bulletin No. 370, U. S. Geological Survey.) By R. L. Humphrey. Washington, U. S. Government, 1909. (Exchange.)

LA FONDERIE MODERNE. Year 10, No. 5–date. Charleville, 1909–date. (New exchange.)

FORMULES, TABLES ET RENSEIGNEMENTS USUELS. Ed. 11, Vols. 1–2. By J. Claudel. Paris, H. Dunod et E. Pinat, 1907. (Purchase.)

FORTYMILE QUADRANGLE, YUKON-TANANA REGION, ALASKA. (Bulletin No. 375, U. S. Geological Survey.) By L. M. Prindle. Washington, U. S. Government, 1909. (Exchange.)

GENESIS OF METALLIC ORES AND OF THE ROCKS WHICH ENCLOSE THEM. By Brenton Symons. London, *The Mining Journal.* 1908. (Purchase.)

GEOLOGY OF THE MIKONUI SUBDIVISION, NORTH WESTLAND. (Bulletin No. 6, New Zealand Geological Survey.) By P. G. Morgan. New Zealand, J. Mackay, 1908. (Exchange.)

GILA RIVER ALUMINA PROPERTY, GRANT COUNTY, NEW MEXICO. N. p., n. d. (Gift.)

GOLD: ITS GEOLOGICAL OCCURRENCE AND GEOGRAPHICAL DISTRIBUTION. By J. M. Maclaren. London, *The Mining Journal*, 1908. (Purchase.)

GRAND DICTIONNAIRE UNIVERSEL. By Pierre Larousse. 17 vols. Paris, 1865–78. (Purchase.)

GREAT BRITAIN—MINES AND QUARRIES. General Report, with Statistics for 1908, Pts. I. and IV. London, 1909. (Exchange.)

GUIDE PRATIQUE DU CHIMISTE MÉTALLURGISTE ET DE L'ESSAYEUR. By L. Campredon. Paris, 1909. (Purchase.)

HANDBOOK FOR FIELD GEOLOGISTS. Ed. 2. By C. W. Hayes. New York, J. Wiley & Sons, 1909. Price, $1.50 net. (Gift of Publishers.)

[SECRETARY'S NOTE.—This handbook, based upon one published in 1908 for the use of the members of the U. S. Geological Survey, has already passed to a second edition, and is well known to American geological students and workers. It is praiseworthy both for what it omits and for what it includes. Its forms and methods of observation and record are admirable; and their use by all practitioners will bring fullness, accuracy, and fitness for comparison into field-reports—which have been hitherto annoyingly defective and heterogeneous. The typography is extremely attractive, and Dr. Hayes may be congratulated upon the proof-reading in the main, though perhaps he will permit me to condole with him over p. 10, l. 14, which speaks of "fact, as distinguished from interference." The irresistible inference is that this "interference" was a disagreeable fact!—R. W. R.]

HANDBOOK OF ALASKA. By A. W. Greely. New York, C. Scribner's Sons, 1909. (Exchange.)

ILLINOIS MINERAL PRODUCTION, 1908. By R. S. Blatchley. Urbana, University of Illinois, 1909. (Exchange.)

INDUSTRIE DES MÉTAUX SECONDAIRES ET DES TERRES RARES, By P. Nicolardot. Paris, O. Doin, 1908. (Purchase.)

INSTITUTE OF METALS. Journal. Vol. I. London, 1909. (New exchange.)

INTERNATIONAL CABLE DIRECTORY OF THE WORLD, 1909. New York–London, International Cable Directory Co., 1909. (Gift.)

INTRODUCTION TO THE RARER ELEMENTS. Ed. 2. New York, J. Wiley & Sons, 1908. (Purchase.)

INTRODUCTION TO THE STUDY OF METEORITES. Ed. 10. By L. Fletcher. London, W. Clowes & Sons, 1908. (Gift of British Museum of Natural History.)

INTRODUCTION TO THE STUDY OF ROCKS. Ed. 4. By L. Fletcher. London, 1909. (Gift of British Museum of Natural History.)

INVESTIGATIONS RELATING TO IRON AND MANGANESE BY THE U. S. GEOLOGICAL SURVEY IN 1908. (Advance chapter from Bulletin No. 380, U. S. Geological Survey.) Washington, U. S. Government, 1909. (Exchange.)

IRON AND STEEL. By J. H. Stansbie. New York, D. Van Nostrand Co., 1908. (Purchase.)

ISTHMIAN GEOLOGY AND THE PANAMA CANAL. By Ernest Howe. N. p., n. d. (Gift of Author.)

JAHRBUCH DER KÖNIGLICH PREUSSISCHEN GEOLOGISCHEN LANDESANSTALT UND BERG-AKADEMIE ZU BERLIN. Vol. 26, 1905. Berlin, 1908. (Exchange.)

JAHRES-BERICHT ÜBER DIE LEISTUNGEN DER CHEMISCHEN TECHNOLOGIE. Pt. 2, 1908. Leipzig, 1909. (Purchase.)

LISLE GOLDFIELD. (Bulletin No. 4, Tasmania Geological Survey.) By W. H. Twelvetrees. Hobart, J. Vail, 1909. (Exchange.)

MECHANICAL APPLIANCES OF THE CHEMICAL AND METALLURGICAL INDUSTRIES. By Oskar Nagel. New York, 1908. (Purchase.)

MECHANICAL ENGINEERING OF COLLIERIES. Ed. 2, 2 vols. By C. J. Futers. London, Colliery Guardian Co., 1908–1909. (Purchase.)

DIE METALLFÄRBUNG. Ed. 3. By Georg Buchner. Berlin, M. Krayn, 1906. (Purchase.)

METALLIC ALLOYS; THEIR STRUCTURE AND CONSTITUTION. By G. H. Gulliver. London, C. Griffin & Co., 1908. (Purchase.)

METALLURGICAL CALCULATIONS. Pt. III. By J. W. Richards. New York, McGraw Publishing Co., 1908. (Purchase.)

MICHIGAN COLLEGE OF MINES. Year Book, 1908–1909. Houghton, 1909. (Exchange.)

MINERAL INDUSTRY. Vol. 17. New York, McGraw-Hill Book Company, 1909. (Gift of Publishers.)

MINERAL RESOURCES OF ALASKA, REPORT ON PROGRESS OF INVESTIGATIONS IN 1908. (Bulletin No. 379, U. S. Geological Survey.) By A. H. Brooks, and others. Washington, U. S. Government, 1909. (Exchange.)

MINERAL RESOURCES OF THE KOTSINA-CHITINA REGION, ALASKA. (Bulletin No. 374, U. S. Geological Survey.) By F. H. Moffit and A. G. Maddren. Washington, U. S. Government, 1909. (Exchange.)

MINERS' POCKET-BOOK. Ed. 5. By C. G. Warnford-Lock. London, E. & F. N. Spon, 1908. (Purchase.)

MISSISSIPPI RIVER. Map of the Mississippi River from the Falls of St. Anthony to the Junction of the Missouri River, in 27 sheets compiled from maps made in 1878–1879 by H. Voss and A. J. Stibolt, 1903–1905. Scale, 1 inch to 20 miles. (Gift of Henry G. Granger.)

MODERN PRACTICE IN MINING. Vol. 1. By R. A. S. Redmayne. London, Longmans, Green & Co., 1908. (Purchase.)

MUNICIPAL ENGINEERS OF THE CITY OF NEW YORK. Constitution, By-Laws, List of Members, and Annual Report, 1908. New York, 1909. (Gift.)

NATAL GOVERNMENT RAILWAYS. Report of the General Manager of Railways for 1908. Pietermaritzburg, 1909. (Exchange.)

NEW SOUTH WALES—MINES DEPARTMENT. Annual Report, 1908. Sydney, W. A. Gullick, 1909. (Exchange.)

OFFICIAL BLUE BOOK OF THE JAMESTOWN TER-CENTENNIAL EXPOSITION, A.D. 1907. Norfolk, Colonial Publishing Co., 1909. (Purchase.)

PACIFIC COAST GAS ASSOCIATION. Proceedings of the 13th and 14th Annual Meetings. San Francisco, 1906. (Gift of Pacific Coast Gas Association.)

PANAMA CANAL MAP SECTION, New York *Evening Post*, Feb. 17, 1909. (Gift.)

PANAMA CANAL MUST BE AT SEA LEVEL, FOR THE HONOR OF THE NATION AND THE SAFETY OF OUR COMMERCE. N. p., n. d. (Gift.)

PAN-AMERICAN SCIENTIFIC CONGRESS. Report of the Delegates of the U. S. to the Pan-American Scientific Congress held at Santiago, Chili, Dec. 25, 1908, to Jan. 5, 1909. Washington, 1909. (Gift.)

PENNSYLVANIA GEOLOGICAL SURVEY. Annual Reports, 1 to 6, 1836–1852. By H. D. Rogers. Harrisburg, 1836–1853. (Purchase.)

PENNSYLVANIA GEOLOGICAL SURVEY. Geology of Pennsylvania. By H. D. Rogers. 2 vols. Philadelphia, J. B. Lippincott & Co., 1856. (Purchase.)

PERU. Cuerpo de Ingenerio de Minas. El Antimonio en el Peru. (Boletin No. 68.) Por Eugen Weckwarth. Lima, 1908. (Exchange.)

——— Ya Cimentos. (Boletin No. 69.) Por Erneste du Bois Lukis. Lima, 1909. (Exchange.)

PETIT LAROUSSE ILLUSTRÉ NOUVEAU DICTIONNAIRE ENCYCLOPÉDIQUE. Paris, 1907. (Purchase.)

PETROLEUM AND COALS. By Eugene Coste. Montreal, Canadian Mining Society, 1909. (Gift.)

PHILIPPINE ISLANDS—BUREAU OF SCIENCE. Annual Report, 7th. Manila, 1909. (Exchange.)

PRACTICAL COAL MINING. Vols. 1–4, 6. Edited by W. S. Boulton. London, Gresham Publishing Co., 1907. (Purchase.)

PRINCIPLES OF INORGANIC CHEMISTRY. Ed. 3. By Wilhelm Oswald. London, Macmillan & Co., 1908. (Purchase.)

RAILROADS: VALUATION AND CAPITALIZATION. Compiled under the direction of Hermann H. V. Meyer. Washington, Library of Congress, 1909. (Exchange.)

RAPID METHODS FOR THE CHEMICAL ANALYSIS OF SPECIAL STEELS, STEEL-MAKING ALLOYS, AND GRAPHITE. By C. M. Johnson. New York, J. Wiley & Sons, 1909. (Purchase.)

RECLAMATION RECORD. June, 1909–date. Washington, 1909–date. (Exchange.)

REPORT ON THE EXAMINATION OF SOME IRON ORE DEPOSITS IN THE DISTRICTS OF THUNDER BAY AND RAINY RIVER, PROVINCE OF ONTARIO. By F. Hille. Ottawa, 1908. (Exchange.)

SMOKELESS COMBUSTION OF COAL IN BOILER PLANTS. (Bulletin No. 373, U. S, Geological Survey.) By D. T. Randall and H. W. Weeks. Washington. U. S. Government, 1909. (Exchange.)

SOME DESERT WATERING PLACES IN SOUTHEASTERN CALIFORNIA AND SOUTHWESTERN NEVADA. (Water-Supply Paper No. 224, U. S. Geological Survey.) By W. C. Mendenhall. Washington, U. S. Government, 1909. (Exchange.)

SOUTH DAKOTA SCHOOL OF MINES. Annual Catalogue, 1909–1910. Sioux Falls, 1909. (Exchange.)

——— Book of Views, 1909.

STATISTICAL ABSTRACT OF THE UNITED STATES, 1908. Washington, U. S. Government, 1909. (Exchange.)

STATISTISCHE ZUSAMMENSTELLUNGEN ÜBER BLEI, KUPFER, ZINK, ZINN, ALUMINIUM, NICKEL, QUECKSILBER, UND SILBER. 15 Jahrgang, 1899–1908. Von der Metallgesellschaft, der Metallurgischen Gesellschaft A.-G. Frankfurt-am-Main, 1909. (Gift.)

STORY OF AMERICAN COALS. By W. J. Nicolls. Philadelphia, J. B. Lippincott Co., 1904. (Purchase.)

SUPERVISION OF STREET RAILWAYS IN ENGLAND AND PRUSSIA. Albany, J. B. Lyon Co., 1909. (Gift of Public Service Commission for the First District, State of New York.)

SURVEYORS' INSTITUTION. List of Members, 1909. Westminster, 1909. (Exchange.)

TÄTIGKEIT DER PHYSIKALISCH-TECHNISCHEN REICHSANSTALT IM JAHRE, 1908. (Reprint.) N. p., n. d. (Exchange.)

TECHNICAL METHODS OF CHEMICAL ANALYSIS. Vol. I., Pts. 1–2. By George Lunge. New York, D. Van Nostrand Co., 1908. (Purchase.)

TECHNISCHE AUSKUNFT. Monatschrift des Internationalen Institutes für Techno-Bibliographie. Jan./Feb., 1909. Berlin, 1909. (This publication is the successor to the Repertorium der technischen Journal Literatur, published by the German Patent Office, the last volume of which, for 1908, will be published in the fall of 1909.)

TEXT BOOK OF ASSAYING, FOR THE USE OF THOSE CONNECTED WITH MINES. Ed. 11. By C. and J. J. Beringer. London, C. Griffin & Co., 1908. (Purchase.)

THROUGH THE YUKON AND ALASKA. By T. A. Rickard. San Francisco, *Mining and Scientific Press*, 1909. Price, $2.50 net. (Gift of Publishers.)

[SECRETARY'S NOTE.—Members of the Institute party which went in 1905 from Vancouver *via* Skagway to Dawson, in the Yukon Territory, will find a large part of this volume occupied with a very readable description of what they saw, accompanied with admirable photographic illustrations, which they will welcome as souvenirs of that memorable experience. Mr. Rickard's account of the Treadwell mine, and his popular sketch of the development of mining-methods in the Yukon Territory and at Fairbanks and Nome, are extremely interesting summaries; and his sketches of life, industry, and scenery throughout the wide region traversed by his journey of 8,000 miles betray the eye of a quick observer and the hand of a practiced reporter. Minor faults, such as a little too much "fine writing," and a proneness to slashing condemnation of men and methods, disappear in view of the general value and attractiveness of the book.—R. W. R.]

TIN FIELD OF NORTH DUNDAS. (Bulletin No. 6, Tasmania Geological Survey.) By L. K. Ward. Hobart, J. Vail, 1909. (Exchange.)

U. S. GEOLOGICAL SURVEY. Geologic Folios of the U. S. Nos. 160, 161, 162, 163, 164, 165, 166. Washington, 1909. (Exchange.)

U. S. PATENT OFFICE. Annual Report of the Commissioner of Patents, 1908. Washington, 1909. (Exchange.)

UNIVERSAL ENGLISH AND GERMAN-ENGLISH DICTIONARY. 2 parts in 3 vols. By Felix Flügel. Ed. 4. Braunschweig, G. Westermann, 1894, 1908. (Purchase.)

UNIVERSITY OF PENNSYLVANIA. Proceedings of Commencement, June 16, 1909. Philadelphia, 1909. (Exchange.)

VEREIN DEUTSCHER CHEMIKER, E. V. Mitglieder-Verzeichnis für 1909. Leipzig, 1909. (Exchange.)

VEREIN DEUTSCHER INGENIEURE, Mitgliederverzeichnis, 1909. Berlin, 1909. (Exchange.)

WATER-SUPPLY INVESTIGATIONS IN THE YUKON-TANANA REGION, ALASKA, 1907 AND 1908. Fairbanks, Circle, and Rampart Districts. (Water-Supply Paper No. 228, U. S. Geological Survey.) By C. C. Covert and C. E. Ellsworth. Washington, U. S. Government, 1909. (Exchange.)

WEST VIRGINIA—DEPARTMENT OF MINES. Annual Report, 1908. Charleston, Tribune Printing Co., 1909. (Exchange.)

WISCONSIN RAILROAD COMMISSION. Annual Report, 1st. Madison, Democrat Printing Co., 1908. (Gift of Railroad Commission of Wisconsin.)

——— Biennial Report, 1st. Madison, Democrat Printing Co., 1907. (Gift of Railroad Commission of Wisconsin.)

WORLD'S GOLD. Its Geology, Extraction, and Political Economy. By L. DeLaunay. New York, G. P. Putnam's Sons, 1908. (Purchase.)

TRADE CATALOGUES.

DENVER ROCK DRILL & MACHINERY CO., Denver, Colo. Detail description, parts, uses, advantages, and specifications of "Waugh" Drills for stoping, drifting, and sinking.

FOOTE MINERAL CO., Philadelphia, Pa. Offers for small mineral collections either for cash or exchange by the above company.

GENERAL ELECTRIC CO., Schenectady, N. Y.

Bulletin No. 4661, May, 1909. Aluminum Lightning-Arresters for Alternating-Current Circuits, showing formation of film and film-solution in aluminum cells, construction of arresters, transfer device, installation, discharge-alarms, choke-coils.

Bulletin No. 4662, May, 1909. Detail drawings, construction, advantages of the Thomson Recording Wattmeters for Switchboard Service.

Bulletin No. 4663, May, 1909. The designing, construction, and detail description of Multigap Lightning-Arresters for Alternating Currents.

Bulletin No. 4669, May, 1909. Curtis Steam-Turbines for Low-Pressure and Mixed Pressure. I. In connection with engines which are run non-condensing. II. In connection with condensing engines.

Telephone Line Insulating Transformer, its construction and advantages.

Series Tungsten Lighting. By Henry Schroeder. Contrasting cost of power and operation of tungsten and carbon lamps.

Supply Catalogue No. 4645, March, 1909. A catalogue containing specifications and prices of General Electric Supplies. Some of the listed articles are arc-lights, lighting-systems, measuring-instruments, transformers, switchboards, motors, panel-boards, fuses, cut-outs, air-brake and oil switches, wires and cables.

GREENE, TWEED & Co., 109 Duane St., New York, N. Y. Cuts, details, parts, construction, and advantages of latest improved Rochester Automatic Lubricators for all styles of engines.

JEFFREY MANUFACTURING Co., Columbus, Ohio. Bulletin No. 18A, June, 1909. Cuts and descriptions of the Jeffrey Air-Power Coal-Cutter and its use in mines.

J. GEO. LEYNER ENGINEERING Co., Littleton, Colo.

Leyner Bulletin No. 1003, June 25, 1909. Information concerning the water and air-drills made by the company.

Bulletin No. 1005, June 10, 1909. Reprint: Comparative Tests of Large and Small Hammer Rock-Drills.

D'OLIER ENGINEERING Co., Philadelphia, Pa. Article reprinted from the *American Sugar Industry and Beet Sugar Gazette:* Garden City Project of the United States Reclamation Service for the purpose of irrigating a strip of land near Garden City, Kansas.

ORE CONCENTRATION Co., London, England.

Working-costs of the Elmore Vacuum Process at various mines.

Testimonials from mines using the Elmore Vacuum Process as regards its efficiency.

Elmore Vacuum Process at Broken Hill, New South Wales. By Stanley Elmore.

Description of the Vacuum Flotation Process for Concentration. By A. S. Elmore.

Reprint from *Mining Journal:* Ore Reduction at the Telemarken Copper-Mine, Norway. By W. E. Bennett.

SIMON-CARVES, Manchester, England. Cuts of some modern By-Product Coke-Oven and Coal-Washing Plants.

WOOD DRILL WORKS, Paterson, N. J. Glimpses of the Panama Canal, with special reference to the work done by the power-drills of above company.

C. O. BARTLETT & SNOW Co., Cleveland, Ohio. Catalogue No. 25, 1908 edition, of coal-crushers, crushing-rolls, clay-disintegrators, ore-crushers, mixing-cylinders, stone-mills, rotary- and automatic-feeders.

YAWMAN & ERBE MFG. Co., Rochester, N. Y. Cuts and descriptions of Convention Record Systems for Filing and Reference Work.

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members or associates who accepted election during the month of July, 1909:

Members.

David R. C. Brown,	Aspen, Colo.
Henry L. Browne,	Swansea, Ariz.
Ernest F. Burchard,	Washington, D. C.
Herbert T. Burls,	London, England.
Cipriano R. Careaga,	Bilbao, Spain.
Alexander J. F. Crauford,	Silver City, N. M.
Erle V. Daveler,	Tonopah, Nev.
John Deegan,	Los Angeles, Cal.
Wallace F. Disbrow,	Hazel Green, Wis.
George S. Evans,	Silver City, N. M.
Carlton E. Fortney,	Coahuila, Mex.
John G. G. George,	Nacozari, Sonora, Mex.
Charles C. Gressang,	Wright, W. Va.
Ernest Howe,	Newport, R. I.
Frank W. Iredell,	New York, N. Y.
William B. McPherson,	Los Angeles, Cal.
Frank S. Mills,	Andover, Mass.
Ledlie D. Moore,	Santiago de Cuba, Cuba.
Andrew W. Newberry,	Kelvin, Ariz.
Abner A. Osborn,	Parryville, Pa.
Morris B. Parker,	El Paso, Texas.
G. Clinton Ripley,	Craft, Cal.
Mortimer F. Sayre,	Bisbee, Ariz.
Joseph S. Shaw,	Algoma, W. Va.
Edgar K. Soper,	Ithaca, N. Y.
Chevalier B. Staples,	Cranbrook, B. C., Can.
Jesse A. Stewart,	McKinley, Minn.
Joseph S. Stringham,	Detroit, Mich.
Fremont N. Turgeon,	Firmeza, Cuba.
Lawrence H. Underwood,	Wheeling, W. Va.
Chester W. Washburne,	Washington, D. C.
Albert G. Wolf,	Telluride, Colo.
Richard Ziesing,	Cleveland, Ohio.

Associates.

Haakon A. Berg,	Midland, Pa.
Roger W. Newberry,	New Haven, Conn.
Henry De Witt Smith,	New London, Conn.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed for election as members of the Institute during the month of July, 1909. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

Henry Truman Beckwith,	Philadelphia, Pa.
William B. Foote,	Geneva, N. Y.
Nathaniel Grant,	Kansas City, Mo.
Thomas Higgins,	Panulcillo, Chile, S. America.
Cleaveland Hilson	Electric, Mont.
Frederick Charles Holmes,	Firmeza, Cuba.
Clarence Victor Hopkins,	Jerome, Ariz.
Warren Lawson Kluttz,	Thomas, Ala.
Robert Gilliam Lassiter,	Virgilina, Va.
Robert Livingstone,	Edmonton, Alberta, Can.
James P. McCarthy,	Wallace, Idaho.
Duncan M. Munro,	Hostotipaquillo, Jalisco, Mex.
John O. Norbom,	Berkeley, Cal.
Jesse C. Porter,	Brooklyn, N. Y.
Guy Hall Ruggles,	Kelvin, Ariz.
William Wearne,	Hibbing, Minn.

Change of Status.

Paul H. Mayer,	Denver, Colo.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of July, 1909. This list, together with the lists given in the *Bulletin* for February, March, April, May, June, and July, therefore, supplements the annual list of members corrected to Jan. 1, 1909, and brings it up to the date of Aug. 1, 1909. The names of Members who have accepted election during the month (new members), are printed in *italics*.

AHBE, FREDERICK R.....................................615 South Main St., Athens, Pa.
ANDERSON, AXEL E., E. I. Du Pont Co....527 Du Pont Block, Wilmington, Del.
ASTLEY, JOHN W., Care R. F. Segsworth.....103 Bay St., Toronto, Ontario, Can.
BANKS, HARRY P..95 Yesler Wy., Seattle, Wash.
BARBOUR, PERCY E., General Delivery..........................Salt Lake City, Utah.
BEL, J. MARC, Expurt des le Conseil de Prefecture de la Seine,
73 Boulevard St. Michel, Paris, France.
†*Berg, Haakon A.*, Blast Furn. Supt., Midland Steel Co.........Midland, Pa. '09.
BLAKE, D. E., Supt., San Mauricio Gold Mining Co............Isd. of Luzon, P. I.
BONILLAS, YGNACIO..P. O. Box 154, Nogales, Ariz.
**Bradley, Richard J. H.*, Min. Eng...........52 Broadway, New York, N. Y. '02.
BRAYTON, COREY C.................. ..Room 325, Clumie Bldg., San Francisco, Cal.
BREWER, ARTHUR K..............................1755 Clement St., San Francisco, Cal.
BROCKUNIER, S. H..................................Pleasant Valley, Wheeling, W. Va.
BROWN, AUSTIN H.....................................2747 Alcatraz Ave., Berkeley, Cal.
**Brown, David R. C.*, Mining Operator................................Aspen, Colo. '09.
BROWN, JOSEPHUS J., JR., Oklahoma School of Mines.............Wilburton, Okla.
**Browne, Henry L.*, Supt. of Mines..................................Swansea, Ariz. '09.
**Burchard, Ernest F.*, Economic Geologist, U. S. Geological Survey,
Washington, D. C. '09.
**Burls, Herbert T.*, Cons. Engr., Care Harbord & Riley,
16 Victoria St., London, S. W., England.
**Careaga, Cipriano R.*, Genl. Mgr., Sociedad Espanola Minas del Castillo
de las Guardas, Bilbao, Spain. '09.
CHANEY, R. GORDON, JR..... ....Goldfield, Nev.
CHAPMAN, MELVILLE D................................80 Broadway, New York, N. Y.
COLLINS, FRANCIS W....................................15 William St., New York, N. Y.
COLLINS, GLENVILLE A............................307 First Ave., South, Seattle, Wash.
**Craufurd, Alexander J. F.*, Cons. Mining Engr., Box D, Silver City,
New Mexico. '09.
CROWLEY, T. IRWIN.....................................16 College Green, Dublin, Ireland.
**Daveler, Erle V.*, Milling..........................P. O. Box 402, Tonopah, Nev. '09.
DAWBARN, GILBERT J., Min. Engr............................Wallaroo, So. Australia.
**Deegan, John*, Mine Manager.....................508 Lissner Bldg., Los Angeles, Cal.
**Disbrow, Wallace F.*, Min. Engr., Mgr. Kennedy Mine, Hazel Green, Wis. '09.
DOUGLASS, ROSS E.............................2330 La Mirada Ave., Los Angeles, Cal.
DURELL, CHARLES T..............................415 O'Farrell St., San Francisco, Cal.
EASLEY, GEORGE R. D.............. ....P. O. Box 953, Baker City, Ore.
EASTON, HARRY D..............................430 Transylvania Park, Lexington, Ky.
EDWARDS, WILLIAM S......................................110 E. 14th St., Tucson, Ariz.
ENFELHARDT, ERNEST C., Mary Murphy G. M. Co....................St. Elmo, Colo.
**Evans, George S.*, Consulting Mining Engr........Box D, Silver City, N. M. '09.
EVANS, S. G ....64 Ashland Ave., East Orange, N. J.
FORBES, DONALD G.........................Shillingstone, Blandford, Dorset, England.
**Fortney, Carlton E.*, Mining Engineer, Cia Carbonera de C. P. Diaz,
Coahuila, Mexico. '09.
FOX, ARTHUR C., Gen. Supt., Duluth-Pacific Copper Co.............Duluth, Minn.
FRANKE, EMIL A., General Supt., Tinton Reduction Co.,
Tinton, Lawrence Co., S. D.
FULTON, THOMAS T., Supt. for Canada Iron Corp., Ltd......Bathurst, N. B., Can.
**George, John G. G.*, Min. Engr.........................Nacozari, Son., Mexico. '09.
GLEASON, FRANK A.....................................1916 Capouse Ave., Scranton, Pa.

GLEASON, WALTER G..P. O. Box 887, Baker City, Ore.
**Gressang, Charles C.*, Supt. of Mines..................................Wright, W. Va. '09
GRIFFIN, FITZ ROY N., Charterlands Goldfields, Ltd.,
Box 468, Bulawayo, Rhodesia, South Africa.
HANCKEL, CHRISTIAN.........71 Kremlin Drive, Stoneycroft, Liverpool, England.
HEARNE, JULIAN..................................27 City Bank Bldg., Wheeling, W. Va.
HENDERSON, ENOCH..Vicksburg, Ariz.
HINMAN, B. C......................Coventry Ho., South Pl., London, E. C., England.
HOLBERTON, WALTER T....................................Copiapo, Chile, South America.
HOLLISTER, JOHN J...................................Santa Barbara, Chihuahua, Mexico.
**Howe, Ernest*, Geol...75 Kay St., Newport, R. I. '09.
HUNDESHAGNE, LUDWIG.......................354 Marnix Str., Amsterdam, Holland.
**Iredell, Frank W.*, Mgr., C. & G. Copper Co., 11 Broadway, New York, N. Y. '09.
JACKSON, JOHN F..............................505 Colby-Abbot Bldg., Milwaukee, Wis.
JANEWAY, JOHN H., JR.......................................55 Wall St., New York, N. Y.
JOHNSON, J. E., JR., General Supt., Thomas Division,
Republic Iron & Steel Co., Thomas, Ala.
KEMPTON, C. W..42 Broadway, New York, N. Y.
KING, AUSTIN J..............................1545 Washington St., Charleston, W. Va.
KING, L. M...1339 Grove St., Alameda, Cal.
KIRKCALDY, NORMAN M., Mussin, Hamitoff & Co......Ust Kaminegorsk, Siberia.
KURYLA, MICHAEL H., Metallurgical Engineer,
Care Esperanza Mining Co, El Oro, Mexico, Mexico.
LEWIS, H. M., JR..........................122 E. South Temple, Salt Lake City, Utah.
McCARTHY, EDWARD T.............125 Victoria St., Westminster, S. W., England.
McCLAVE, JAMES M..617 Mercantile Bldg., Denver, Colo.
**McPherson, William B.*, Mine Manager, 415½ South Spring St.,
Los Angeles, Cal. '09.
MACNUTT, C. H., Care Beazley & Co..................Antofagasta, Chile, So. Amer.
MAYER, PAUL H..1425 Washington St., Denver, Colo.
**Mills, Frank S.*, Min. Geol.......................P. O. Box 644, Andover, Mass. '09.
**Moore, Ledlie D.*, Mining Engineer.......Firmeza, Santiago de Cuba, Cuba. '09.
MORRIS, ROBERT H.......................1510 Brown-Marx Bldg., Birmingham, Ala.
NACK, CHARLES...............................14 Augusta, Karlsruhe, Baden, Germany.
**Newberry, Andrew W.*, Chemist, Ray Cons. Copper Co......... Kelvin, Ariz. '09.
†*Newberry, Roger W.*, Student..................137 Wall St., New Haven, Conn. '09.
NICOLL, BENJAMIN ....149 Broadway, New York, N. Y.
NYE, ROBERT, Min. Engr...Grass Valley, Cal.
**Osborn, Abner A.*, Chem......Care Carbon Iron & Steel Co., Parryville, Pa. '09.
**Parker, Morris B.*, Min. Engr...................P. O. Box 52, El Paso, Texas. '09.
PATTBERG, OTTO F., St. Lawrence Pyrites Co.....................De Kalb Jc., N. Y.
PLUEMER, A..231 Murray St., Elizabeth, N. J.
RAYMOND, P. A...Instructed to hold all mail.
REINHOLT, OSCAR H...................................555 So. Hope St., Los Angeles, Cal.
REYNOLDS, LLEWLLYN..................................Socorro Mines, Mogollon, N. M.
RICE, GEORGE S., U. S. Geological Survey...40th and Butler Sts., Pittsburg, Pa.
**Ripley, G. Clinton*, Tunnel Superintendent, Los Angeles Aqueduct,
Craft, Cal. '09.
ROBERTSON, J. D.................Las Menores, via San Juan del Rio, Dur., Mexico.
RUETSCHI, RUDOLF ....10 Fort Greene Place, Brooklyn, N. Y.
**Sayre, Mortimer F.*, Mining Engineer, Copper Queen Cons. Mining Co.,
Box 2127, Bisbee, Ariz. '09.

SHARWOOD, W. J. .... Lead, S. D.
*Shaw, Joseph S., Chief Engr., Coal & Coke Co. .... Algoma, W. Va. '09.
SHOEMAKER, G. M. .... P. O. Box 160, Huntington, W. Va.
SMALL, HARVEY B., Min. Engr., Care Empresa Hauseatica, Barranquilla, Colombia, So. Amer.
SMITH, FREDERICK D., Mgr., Seven Devils Copper Co. .... Landore, Idaho.
†Smith, Henry De W., Mining Student, 1080 Ocean Ave., New London, Conn. '09.
SMITH, NOAH B., Smith, Rudy & Co. .... 411 Walnut St., Philadelphia, Pa.
*Soper, Edgar K., Instructor, Economic Geology, Cornell University, Ithaca, N. Y. '09.
SPILSBURY, PERSIFOR G. .... 45 Broadway, New York, N. Y.
STANTON, WILLIAM A. .... 811 Trinity Bldg., New York, N. Y.
*Staples, Chevalier B., Mining Engineer, P. O. Box 236, Cranbrook, B. C., Canada. '09.
*Stewart, Jesse A., Mining Engineer, Elba Mine .... McKinley, Minn. '09.
STODDARD, A. W. .... 638 Salisbury House, London, E. C., England.
*Stringham, Joseph S., Mech. Engr., Solvay Process Co. .... Detroit, Mich. '09.
THOMAS, DAVID R., Manager, Predilecta Mining Co., Guanacevi, Durango, Mexico.
THROPP, JOSEPH E., JR. .... Mercersburg, Pa.
TOD, STUART .... Care Marwick, Mitchell & Co., 79 Wall St., New York, N. Y.
*Turgeon, Fremont N., Min. Engr., Juragua Iron Co. .... Firmeza, Cuba. '09.
*Underwood, Lawrence H., Supt. Blast Furn., Riverside Wks., National Tube Co., Wheeling, W. Va. '09.
VALENTINE, CHARLES F. .... Emuford, Cairns, No. Queensland, Australia.
WALKER, MYRON R. .... Apartado 145, Oaxaca, Oaxaca, Mexico.
*Washburne, Chester W., Geologist, Off. U. S. Geological Survey, Washington, D. C. '09.
WEBSTER, P. W., Treadwell Construction Co. .... Midland, Pa.
WELLS, JAMES S. C. .... 320 W. 83d St., New York, N. Y.
WEST, HAARLEM E., El Oro Mining & Railway Co. .... El Oro, Mexico, Mexico.
WHITE, EDWIN E., Cleveland Cliffs Iron Co. .... Ishpeming, Mich.
WHITWORTH, SAMUEL .... Equitable Bldg., Melbourne, Vic., Australia.
WILSON, C. HERBERT .... Jubbulpore, Central Province, India.
*Wolf, Albert G., Mining Engineer, Smugglers Union Mining Co., Telluride, Colo. '09.
*Ziesing, Richard, Mgr. Spelter Plant, 8th floor, Old Arcade, Cleveland, Ohio. '09.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address on Records, from which Mail has been Returned.
Adams, Randolph,	Copperhill, Tenn.
Alexander, George E.,	Sparta, Ore.
Allen, Frederick E.,	Bloomsburg, Pa.
Andrew, Thomas,	Pretoria, So. Africa.
Arozarena, R. M. de,	Mexico City, Mexico.
Atkin, Austin J. R.,	Steynsdorp, So. Africa.
Bartoccini, Astolfo,	214 E. 90th St., New York, N. Y.
Bassett, Thomas B.,	Cumpas, Sonora, Mexico.
Batchelder, Joseph F.,	54 1st St., Portland, Ore.
Bellam, Henry L.,	Reno, Nev.
Bouchelle, James F.,	22 Duncan Ave., Jersey City, N. J.
Brook, Henry E. C.,	Cadia, N. S. W., Australia.

Brown, Frank H., ... Coppermount, Alaska.
Campa, Jose, ... Mexico City, Mexico.
Collins, W. J., ... Johannesburg, So. Africa.
Cragoe, A. Spencer, ... Vencedora, Mexico.
Derby, Harry S., ... 134 Monroe St., Chicago, Ill.
Dougherty, Clarence E., ... 41 Wall St., New York, N. Y.
Ekberg, Benjamin P., ... Johannesburg, Transvaal, So. Africa.
Field, Wilfrid B., ... Mexico City, Mexico.
Fitzsimmons, F. J., ... Cananea, Mexico.
Francis, George G., ... 177 St. George's Sq., London, W., England.
Fuller, Frederick D., ... Sumpter, Ore.
Gage, Edward C., ... San Dimas, Dur., Mex.
Gee, Emerson, ... Reno, Nev.
Hunt, Thatcher R., ... Iron Mt., via Keswick, Cal.
Jackson, Byron N., ... Milton, Cal.
Jessop, Herbert J., ... Guanacevi, Mexico.
Jewett, Eliot C., ... 2918 Morgan St., St. Louis, Mo.
Judd, Henry A., ... Mertondale, W. Australia.
Kow, Tong Sing, ... Shanghai, China.
Mildon, Reginald B., ... Nacozari, Son., Mexico.
Moulton, Herbert G., ... Cobalt, Ont., Can.
Muir, Thomas K., ... Portland, Ore.
Nawatny, William F., ... Harrisburg, Ill.
O'Byrne, Joseph F., ... Midas, Nev.
Philbrick, Arthur, ... Manhattan, Nev.
Piper, John W. H., ... Buenos Ayres, Argentine Rep., S. A.
Potter, J. A., ... 41 W. 124th St., New York, N. Y.
Rigney, Thomas P., ... Reno, Nev.
Rodda, Richard W., ... Seattle, Wash.
Sandifer, Harmer C., ... El Oro, Mexico.
Schlemm, William H., ... Durango, Mexico.
Scott, Winfield G., ... Long Beach, Cal.
Skelding, Joseph F., ... Embreeville, Tenn.
Thomas, Richard A., ... 43 Wall St., New York, N. Y.
Vaux, Charles A., ... P. O. Box 80, East Rand, So. Africa.
Vidler, Louis W., ... Lookout Mountain, Colo.
Warren, Henry L. J., ... Salt Lake City, Utah.
Wiswell, Herbert J., ... Carterville, Mo.
Wolfe, Burton L., ... Ely, Nev.
Young, William, ... Kenora, Ont., Canada.

NECROLOGY.

The deaths of the following members have been reported to the Secretary's office during the month of July, 1909:

Date of Election.	Name.	Date of Decease.
1906.	**Robert S. Brooks,	July 6, 1909.
1876.	*Jerome Keeley,	July 4, 1909.
1881.	*Robert Pitcairn,	July 25, 1909.
1906.	*James Stirling,	June 26, 1909.
1871.	**T. F. Witherbee,	July 11, 1909.

* Member. ** Life Member.

BIOGRAPHICAL NOTICES.

Jerome Keeley was born Jan. 9, 1844, at Phœnixville, Pa. His father was a constructing engineer and manager of iron blast-furnaces, and for a considerable period head of the construction-department of the Phœnix Iron Co. It was natural that the son should be disposed by his father's example to choose the same profession; and as soon as he left school he began to work under his father's superintendency. But in January, 1860, he entered the Polytechnic College of the State of Pennsylvania, where he was graduated as mechanical and metallurgical engineer in June, 1862. After graduation, he was engaged as superintendent of three of the blast-furnaces of the Phœnix Iron Co. In 1869, he established an office in Philadelphia as consulting engineer in metallurgy and mining, and also for the handling of iron and steel products, tin-plate, and metals. He was also interested in the manufacture of pig-iron, blooms, bars, etc., and later in the production of coal and coke. In 1895, he was elected Director and Vice-President of the Sheffield Coal, Iron & Steel Co., of Sheffield, Ala., a position which he subsequently resigned, returning to his business in Philadelphia. He was also President of the Durham Iron Co., and held other positions of trust. About six years ago, he was obliged by ill-health to retire from active business, and he died at his home in Philadelphia on July 4. Mr. Keeley joined the Institute in 1876, the year of the Centennial Exposition, which brought the young society into touch with hundreds of engineers at home and abroad, and established its reputation. He gave it his loyal support for thirty-three years.

Thomas F. Witherbee was born Oct. 10, 1843, at Port Henry, N. Y. After receiving a common-school education, he entered the Rensselaer Polytechnic Institute, at Troy, N. Y., but cut short his professional course in 1862, when he entered the Union army.

After receiving, in 1864, his honorable discharge, he went to Fletcherville, N. Y., where he became assistant to J. B. Bailey, superintendent of Witherbee & Fletcher's new charcoal blast-furnace. In a short time he became superintendent, and remained for seven years in that position. Mr. Witherbee became a member of the American Institute of Mining Engineers

in 1871, the year of its organization, at which time he was already known by his work at Fletcherville as a remarkably skillful, energetic, and progressive furnace-manager. One of his earliest innovations at that place was the installation of a laboratory for the analysis of ores and fluxes. He is believed to have been the first manager in America to use the chemical laboratory for the purpose of controlling the regular running of the blast-furnace. His Fletcherville furnace was one of the first in this country to furnish pig-iron for the Bessemer converter.

In 1872, he began at Port Henry the construction of the Cedar Point blast-furnace, in which his brother, Jonathan G., and his uncle, Silas H. Witherbee, were large stockholders, and which was completed under his management, though, by reason of the general financial depression of 1873 and 1874, not put in blast until 1875. Mr. Witherbee remained in charge of this furnace for about seventeen years, and achieved in connection with its management a high reputation. He introduced at the Cedar Point furnace the first Whitwell fire-brick stoves ever erected in this country, importing from England both the fire-brick and the castings. It was here, likewise, that he introduced the Witherbee bronze tuyeres (his own invention). He was the first in America to use the Lürmann water-cooled cinder-notch; the first to remove scaffolds and other obstructions in the blast-furnace with dynamite (whence came the *sobriquet* "Dynamite Tom," often applied to him); and he invented and used the "kerosene blow-pipe" for opening closed tuyeres and tapping-holes.

In 1887, Mr. Witherbee was called to the superintendency of the rich iron-mines and large iron-works of Iron Mountain, Durango, Mex. In 1895, he took charge for a time of the Calumet furnace, Chicago, and for four years thereafter he managed successfully a blast-furnace at Mayville, Wis.

He was then induced to return to his previous work in Durango, where he possessed important private interests also, and where he remained until his death, which took place July 11, 1909. He had suffered a stroke of paralysis March 13, followed June 16 by a second, after which his life ebbed gradually away. For years past he had been almost blind, but this had not prevented him from energetic activity, or even from authorship.

He was one of the earliest members of the Institute, and showed his permanent interest in it, not only by making himself a life-member, but by contributing to the *Transactions* a most varied and valuable series of professional papers, of which the following is a list:

Year.		Vol.	Page.
1873.	The Manufacture of Bessemer Pig-Metal at the Fletcherville Charcoal Furnace, near Mineville, Essex Co., N. Y.,	II.	65
1876.	The Cedar Point Iron Company's Furnace No. 1, at Port Henry, Essex Co., N. Y.,	IV.	369
1876.	Discussion on the Hot-Blast,	V.	79
1877.	Heat Requirement and Gas Analysis at Cedar Point Furnace, Port Henry, N. Y.,	V.	618
1877.	Fluxing Siliceous Iron-Ores,	VI.	164
1877.	A New Method of Taking Blast-Furnace Sections,	VI.	170
1879.	The Working of Three Hearths at the Cedar Point Furnace, Port Henry, N. Y.,	VIII.	34
1880.	Notes on Two Scaffolds at the Cedar Point Furnace,	IX.	41
1881.	The Use of High Explosives in the Blast-Furnace,	X.	206
1885.	Removing Obstructions from Blast-Furnace Hearths and Boshes,	XIII.	675
1901.	Discussion on the Constitution of Cast-Iron,	XXXI.	992, 997
1901.	The Iron Mountain, and the Plant of the Mexican National Iron & Steel Co., Durango, Mexico,	XXXII.	156
1904.	Special Forms of Blast-Furnace Charging-Apparatus,	XXXV.	575
1904.	Discussion on Stock-Distribution and Its Relation to the Life of a Blast-Furnace Lining,	XXXV.	1000
1907.	Blast-Furnace Practice,	XXXVIII.	887

It will be seen that the foregoing list of contributions covers practically the whole of Mr. Witherbee's professional career. Whatever he learned he was willing to impart, and he left for his colleagues and successors the records of his own experience —an example worthy of imitation, and one which now brings its deserved reward; for, besides winning the love and esteem of his contemporaries by his fraternal freedom of communication in technical matters, Mr. Witherbee has left his name in the history of his profession as one of the leaders in the epoch which witnessed the reconstruction of its means, methods, materials, and theories.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Need of Instrumental Surveying in Practical Geology.

BY BENJAMIN SMITH LYMAN, PHILADELPHIA, PA.

(Spokane Meeting, September, 1909.)

THERE seems to be dire need of repeated preachment against the too-frequent sad neglect of instrumental surveying and mapping in geological surveys. The value of the map as an illustration of the statements and opinions of a report is too apt to be overlooked; and its essential necessity in working out during its construction the proper conclusions from the observed facts is generally altogether misunderstood.

Practical geology seeks, of course, to ascertain and indicate the character of workable or unworkable beds or veins, their depth, position, dip, and horizontal course, or strike, even below the surface; also their outcrops, and therefore, if workable, their extent; and so, taking account of their thickness and specific gravity, their weight in tons. It might, therefore, be called quantitative geology. For the study of more abstruse geological questions, too, it is in many cases necessary to gain some knowledge, or reasonable opinion, in regard to such hidden facts. In some cases, careful instrumental surveys are made; in others, from necessity or choice, there are few or no instrumental observations.

It was said, 15 or 20 years ago, that there was an engineer in the anthracite-regions who was capable of telling, on first glancing at a new place, exactly what coal-bed was to be found there at a certain depth, and how thick—a splendid second-sight! "'O, there be,'" you may be tempted to exclaim, "geologists 'that I have seen' geologize 'and heard others praise, and that highly, not to speak it profanely,'"—but let them pass without being fully characterized, lest the temptation towards profanity be altogether too strong. Yet such second-sight does not exceed the expectations of many men of imperfect geological knowledge, men who have heard of marvelously successful geological predictions, but are unaware of

the methods by which they were arrived at. The marvel in such cases, however, is not seldom much exaggerated, and is sometimes altogether imaginary. On one occasion my Japanese assistants gleefully told of a joyful oil-well digger who declared that I had dramatically stamped my foot on the ground at a certain spot and told him to dig there for oil; he had done so, and had been very successful. The story at first seemed wholly a mistake, but, under reflection, proved to have the foundation that he had been told that at a point indicated, a few yards from where he was digging at the outcrop of a steeply-dipping oil-bearing bed, he would find the same bed at a certain depth and probably less drained of its oil. There had been, after all, nothing magical about it, nothing whatever smacking of the divining-rod. It is true that, with or without any such implement or paraphernalia of any kind, but perhaps with much undisplayed knowledge of the subject and of kindred circumstances, and with keen, yet instantaneous, "lightning-calculating" observation of the conditions present, shrewd estimates and sagacious guesses are sometimes made; and this may account for the successes of some of the divining-rod men. The public (including, too, men like deep-hole drillers), somewhat acquainted with underground work, surprised, as they are, at the frequent accuracy of underground prognostications arrived at by instrumental surveying, and considering it to be accomplished only by a sort of magical second-sight, or at least by sagacity, come to expect real second-sight of geologists, and regard such success as merely a matter of course. The results of patient geological investigation are, therefore, apt to be considered mere guesswork, or an ordinary, though mysterious, second-sight; and there is rarely any due appreciation of their real value. A multi-millionaire capitalist, thinking of employing you on a coal-land survey, will ask you half-seriously whether you are able to tell (off-hand, of course) what is 5 fathoms deep below your feet.

But boldly positive and tempting as the declarations of a second-sight man are sometimes made to appear, it is very unsafe, in a case of the least difficulty, to pin your faith upon them without having some satisfactory, rational explanation of their foundation; for they may be grossly misleading. For example, an ore-vein, or a coal-bed, or a set of coal-beds exposed at

one place, may, at a guess, seem, for strong resemblance, or a slight divergence in its course or its character, to be the same or not the same vein, bed, or set of beds as one exposed a quarter or a half a mile away. To take a particular instance, near Schooner pond, on the sea-coast of the principal Cape Breton coal-field, a 3-ft. coal-bed exposed (in 1864) near sea-level appeared possibly to be a different bed from one worked about half a mile away, and again another more than a mile beyond, on the shores of a small headland. An instrumental survey connecting the three points showed them to be perfectly in one straight line and clearly to be upon a single bed, in that region of remarkable uniformity of geological structure. The uniformity, indeed, is so great that the 8-ft. Phalen (or Campbell) coal-bed was, in 1863, successfully opened up on the north side of Big Glace bay, within a few feet of the place indicated by mere instrumental survey from openings about a mile and two-thirds distant on the south side of the bay; and in 1866 was opened further to the northwest at the Caledonia mines by a shaft, where instrumental surveying from the nearest exposure of the bed, three-quarters of a mile distant, indicated that its depth would be 180 ft., and it was found to be 182 ft. But opinions based upon observations without instruments equally suppose the uniformity of the unexposed geological structure to be completely perfect. Yet, even in a case of such uniform, regular structure, it would generally be very unsafe to rely implicitly upon the accuracy of surmises, though they should be made by a wonderfully intelligent man, if they are based merely upon unmapped observations, or generally upon observations mapped without indication of diversities of level—that is, without the topographical map of an instrumental survey; for the mapped position of a high point on a dipping, but not vertical, bed must obviously differ from that of any lower point on the same bed, and in identifying the natural exposures as parts of one bed their elevation, as well as dip, must be taken into account.

Nevertheless, many ungeological and excessively impatient land-owners, or mine-speculators, seem to expect only some such miraculous, yet at the same time accurate, perception of the most hidden facts of a place of supposed mining-promise within half an hour after arriving there by a journey of per-

haps hundreds of miles. Then they are eager to hurry you away and have a report prepared as quickly; or at least to have a preliminary report made without any proper mapping and study of the facts that have been observed. It might be possible in a very plain case, where the beds lie level, or ore-veins are nearly vertical, or where they are evidently quite unworkable; but it is absurd in more complicated cases, with beds of varying dips, and of basin- or saddle-shape, and strongly-curved courses or strikes.

It has already been shown[1] how Lesley played the leading part in bringing topographical mapping to the aid of geological investigations, indicating in 1853 and 1854 the shape of the ground, the hills and valleys, by the more definite and clearer, at that time comparatively novel, contour-lines, instead of the old hachure-lines, with their mountain-ridges looking like caterpillars crawling over the map. The very clearness and definiteness of the contour-lines may seem to require elaborateness and accuracy of instrumental work, but they are also capable of being used advantageously for mere sketching, with indication, of course, that they do not pretend to accuracy.

In 1865 and 1866, a further step forward was taken[2] in indicating the shape of the coal-bed (or other mineral deposit) itself by similar contour-lines, or curves equidistant in level. This, likewise, sometimes seems too precise a method for the uncertain information that may be at hand; but can equally be guarded against being taken as more certain than it really is. The lines are, however, definite, and indicate clearly what is at least supposed to be the geological structure. Like the surface contour-lines, they are a geometrical construction of the shape of the surface to be represented—say, a coal-bed, or other deposit—throughout the area mapped; and the correctness of the structure displayed is severely tested (and perhaps for that very reason the method is less frequently adopted) by its agreement with every known exposure, and by the indication of every possible cross-section, with its series of beds. The method is not only of the greatest value for exhibiting the supposed geological structure, but in the preparation of the map is of yet greater use in working out the most probable

[1] *Trans.*, i., 189–192 (1871–73).
[2] *Trans.*, i., 192 (1871–73); and (J. H. and E. B. Harden) xvi., 290 (1887–88).

structure, by means of numerous trial cross-sections, with the known exposures and their dips and strikes all utilized. Of course, at all the more incomplete stages of the mapping, the tempting second-sight method may be conveniently brought into play, but, of course, too, with uncertain results, which in many cases may be of little value or even seriously misleading. It is true that the complete carrying-out of the instrumental method is laborious and far more time-taking than the field-work, but is well rewarded by the greater certainty of the result. Lesley, the most competent of all judges in the matter, on seeing a few maps made in this way, during his two years' absence in Europe, writes in a letter still extant, June 27, 1868, in warm approval of the method, saying the maps are in "a new style and will probably introduce a new fashion—or rather, I would say, *would* introduce one if there were any well-trained geologists in the country to copy" this "style." But he adds that evidently "it costs enormously in time and brains. I don't object to that myself, you know. And it is the only foundation for a durable reputation." Nearly 20 years later, he, as State geologist, adopted this method of mapping for the great anthracite-survey. The French Geological Survey also has had some of its maps drawn in that way. In Japan many such maps have been made, and a number of them published.

It is true that the instrumental method requires a great deal of time, compared with second-sight. The office-work, if properly done, is so time-taking that our late lamented, able fellow-member, Ellis Clark, formerly assistant on the Pennsylvania State Geological Survey, scarcely exaggerated when he remarked, 20 years ago, that, according to his experience (doubtless without the underground contour-lines), every day of field-work required 10 days of office-work. But a large share of the work, both in the field and in the office, can be done very satisfactorily by a young assistant whose fidelity can be relied on. In the field, he handles the transit, and keeps the survey-notes, with their accompanying sketching, while his chief is free to move about, indicate points to be taken for stations, sketch the topography, and make other observations. In the office, the assistant can, at least, do the plotting, make any needful computations, draw the columnar sections and the cross-sections, do the final tracing, the lettering, and the like. Indeed, a

somewhat well-trained assistant can do all the field-work, except possibly the interpretation of an exceptionally knotty point here and there; and so in the office he could, with a little guidance, do all the work, except the decision of the most important questions. Lesley so worked, especially in regard to the field, which he in some cases would hardly visit at all, sometimes not at all; but he was not averse to the office drudgery, at which he was very adroit, and which he found agreeably reposeful and invigorating, as well as highly conducive to a thorough digestion of the elementary facts.

Of course, instrumental surveying does not always need to be done with instruments of the highest precision. The leveling does not need to be so exact that a polygon will close with an error of only 0.01 ft. Leveling with a pocket-level that closes within a foot or so may be satisfactory, if there be checks to prevent such errors from accumulating, for the rock-beds themselves are more variable than that in thickness. Aneroid leveling, though still less exact, may advantageously be used, if frequently checked by more exact work. For the horizontal work, the large transit-compass is generally precise enough; and even the small prismatic compass is useful, far beyond mere sketching or guessing. Stadia measurement, if without too long sights, is at least as good as chaining; and careful pacing is much better than guessing at distances, and was found by Prof. H. S. Munroe, 35 years ago, in extensive tests of the work of four men, to average in error only about 0.5 per cent. These rough methods are, at any rate, a great improvement upon sketching alone, or second-sight and guess-work, which, to be sure, may be much better than nothing, according to the personal skill and eye-sight of the observer, though apt at times to be grossly erroneous and misleading in spite of the utmost sagacity.

An example of the advantage of merely rough surveying is to be seen in the Pennsylvania State Geological Survey map of the New Red of Bucks and Montgomery counties (1893), where the probable place of outcrop of two important beds of brownstone, valuable for building-purposes, is indicated by two crooked lines running westward many miles through the country from large quarries near the Delaware. The crooks and bends in the lines may by the ignorant be supposed to be mere

fancy work, but were drawn with care according to the height of the ground and the direction and steepness of the dips, turning northward in the low ground, and rising southward in the higher land. Of course, with the unsatisfactory means at hand for so hasty a survey, no very great accuracy could be expected from such indications; and, indeed, nobody apparently put enough faith in them, if at all understanding their meaning, to try with their help to open up the valuable building-stone at any point where it would be convenient to quarry it. Nevertheless, some years after the publication of the map, a quarry in the stone, seemingly discovered by chance, was opened on the marked outcrop at Grenoble, 5 miles west of the nearest old quarry; and another, in the same way, 5 miles still further west, at the trolley power-house. Successes like that in geological mapping are apt, if noticed in any way, to be taken as a matter of course; as if it did not, after all, require any particular intelligence, or common sense, to adopt the so universally neglected instrumental, topographical method, albeit in a field that had for three-quarters of a century been geologically under the unsatisfactory sway of second-sight (or closely kindred) sorts of investigation. Or the successes are apt to be taken as not needing any special care or skill after once adopting a plan so simple, to be sure, in principle, yet so capable of being negligently and inaccurately carried out. In fact, not only ungeological men, but beginners in geology, seem to suppose that such precise indications are altogether cases practically of second-sight, which a geologist is to be expected as a matter of course to possess in a high degree of accuracy—at least as high as any other geologist has. They suppose that "they all do it," and imagine that all that is needed in their own statements of opinion is to "be bold, be bold, be bold," forgetting that they must "be not too bold."

A neglect to complete topographical mapping and to make neighboring cross-sections that would properly take into account the varying course of the rock-beds and their elevation, did in one coal-field lead a highly-sagacious observer to mistake the true identity of the coal-beds and their probable place of outcrop, and to dig in one place a long drift, in another, a deep shaft, in the vain expectation of finding certain coal-beds at points where more thorough mapping and cross-sections later showed

their existence to be clearly quite impossible. A beginner's youthful, but readily pardonable and not unattractive enthusiasm, at another point in the same field, led to his having somewhat extensive fruitless digging done where later the completion of the map showed for the whole neighborhood that such digging would plainly be useless. In another part of the same field certain dips seemed by the second-sight method to indicate great irregularity, as if from numerous and extensive faults; yet a patient investigation with mapping made it clear that no such great disturbance probably existed thereabouts, and that the coal-beds were there likely to be in satisfactorily workable condition. Again, in the same field, before the mapping was completed, two whole sets, each of three or four workable coal-beds, were quite naturally (by second-sight method) supposed to be but one set, though the map later made it sure that they were two distinct sets of beds, overlying one another, and thereby giving to the field a far greater amount of workable coal than had previously been suspected. In such a case a hasty yet not incautious "preliminary report," without waiting for the completion and study of the map and of many trial cross-sections, could hardly fail to be quite erroneous and misleading.

There are countless instances where such geometrical construction of the geological structure has corrected second-sight guesses, and successfully guided the opening-up of coal-beds, visibly and far more satisfactorily than drilling deep holes in the way so fascinating and costly to many men unfamiliar with the capabilities of instrumental surveying. But why multiply special citations?

It is already fully evident that second-sight, tempting as it is for its hare-like celerity, cannot, for certainty of arrival at a satisfactory goal, in the least compare with the invaluable, surer, steady-going, though, if you please, more tortoise-like, process of instrumental surveying. The tortoise is the favorite Japanese emblem of great longevity; and as such might well be applied to the long-lasting useful results of this too-much-neglected method.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Modern Practice of Ore-Sampling.

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(Spokane Meeting, September, 1909.)

FROM the old-fashioned "grab-sample" to the modern timing-device, which takes a machine-sample with mathematical precision, there is a wide gap which was only crossed by many years of toil and unremitting endeavor. Even to-day, notwithstanding the advancement in the art, "grab-sampling" is still practiced—sometimes to afford the unscrupulous mine-promoter a basis for fairy-tales with which to entrap the too-gullible investor, and often by milling and smelting companies to determine the amount of moisture in custom-ores. The latter practice is almost as reprehensible as the former, and it causes more trouble and ill-feeling between seller and buyer than all other factors put together. No reputable concern to-day would think of attempting to determine by grab-sampling the amount of gold, silver, lead, or copper contained in an ore, and yet many buyers expect the miner to accept the results of grab-sampling in the determination of the amount of water contained in the ore, forgetting that accurate results are just as necessary here as in the determination of the metals, because the result determines the percentage of weight of the ore which shall be excluded and considered to have no value whatever.

Samples for the determination of moisture should be taken with as great care as samples for the determination of metallic content, and in order to avoid the extra expense of a separate operation moisture-samples should be taken from the sample-safe. As the sample reaches the sample-bin in a smaller stream and by a more circuitous route than the "reject" travels in its path to the outgoing car, it loses more moisture *en route*, and a constant should be added to compensate for this difference. Carefully-conducted experiments have shown that the difference in loss of moisture between the two routes does not exceed 10 per cent. in summer and 7 per cent. in winter. For instance,

a lot of ore shipped during the summer months, in which the machine-sample showed 5 per cent. of moisture, would have an actual moisture-content of 5.5 per cent. Grab-sampling by an interested party, at its best, is only a prejudiced conjecture, while at its worst it gives rise to the most unscrupulous practices with which the ore-producer and the mining-investor have to deal.

Shovel-sampling, another archaic method which is still used in some localities, consists in throwing out from the car or wagon every third, fifth, or tenth shovelful for a sample. As the portion of the pile from which the sample is taken is entirely at the discretion of the operator, the process would be more properly named fifth-shovel selection than fifth-shovel sampling. Between the conscientious workman who endeavors to be absolutely upright, and often becomes, as the Scotchman said, "maer than plumb," and the scheming laborer who, desiring to make his "job solid," takes a "safe sample," there is little room for truth or accuracy in this method, and the sooner it is consigned to oblivion the better for every one concerned.

Thirty years ago Cornish quartering was the almost universal method of sampling in use, and it is still employed to a considerable extent in cutting-down machine-samples and in mine-examinations where no machinery can be had. When properly carried on with skill, care, and common honesty, fairly-good results may be obtained by quartering, but between the possibility of accidental mistakes and the opportunities which it affords for skillful and unscrupulous operators to manipulate the sample, it has fallen almost into disuse, and should have been completely abandoned long ago. The inherent defect of this system lies in the fact that piling a lot of ore in the form of a cone does not mix it, as the advocates of this system claim. Dropping shovelful after shovelful of ore on top of a cone, instead of building up a homogeneous pile, actually produces a very perceptible sorting-action, whereby the fines build up where they fall on the center of the cone and the coarser particles roll outward and down the sides. This is illustrated in Fig. 1, which is a half-tone from a photograph of a cone built up in actual sampling-practice and bisected by a sheet of glass. This section shows conclusively the great difference in the relative proportion of coarse and fines between the outer and

inner portions of the cone, and also makes it perfectly clear that even after the cone has been spread out into a pancake, as shown in Fig. 2, the fines in the lower portion of the cone will be entirely undisturbed. The most uniform and best results are obtained by coning around a rod, as shown in Fig. 3. By this means the center of the cone is maintained in a vertical line, and if care is taken in working down the cone to a "pancake," as shown in Fig. 2, and separating the quarters by steel blades, so that there is no difference between the quadrants taken for the sample and those thrown into the reject, the results give a fair approximation of the truth, though it is not possible to duplicate results very closely by this method, even at its best.

It would take altogether too much space here to enumerate the different schemes which unprincipled operators have introduced into this method for the purpose of "throwing" the sample, and description of one of them will suffice.

The most ingenious of these plans, and one which is so difficult to detect that it can be carried on directly under the eyes of a skilled observer without detection, is what is known as "drawing the center." The cone is started on the floor, as shown in Fig. 3, but without any rod to determine the position of the center. The operator in charge of the work, in dropping his shovelfuls of ore on the top of the cone, does it in such a manner as to draw the center of the cone imperceptibly in a certain direction, so that by the time the entire sample is piled and ready for spreading, the apex of the cone, shown in Fig. 4, is several inches, we will say, to the SE. of the original center, which is indicated by the perpendicular line, *A*. The ore may now be spread as usual with shovels or with a board, and cut and marked into quadrants by steel blades in alignment with the four points of the compass, as shown in Fig. 5, where the rod, *A*, indicates the original center of the cone, which, of course, has been entirely undisturbed by the mixing and spreading of the upper portion. By rejecting the NW. and SE. quarters an excess ratio of the fines is eliminated, and since these are generally the richest ore the metallic contents of the two retained quadrants, shown in Fig. 5, will be somewhat less than the average of the original pile. Suppose a 2,000-lb. lot is to be reduced to 62.5 lb., it would mean that the "quarter-

ing" (really halving) would have to be repeated five times, and if at each stage the sample taken represented 98 per cent. of the actual value of the cone, the final sample would only give 90.3 per cent. of the true value of the cone, as shown in the following tabulation:

	Original Lot.	First Cut.	Second Cut.	Third Cut.	Fourth Cut.	Fifth Cut.
Weight, lb., ...	2,000	1,000	500	250	125	62.5
Percentage of true value, .	100	98	96	94.1	92.2	90.3

The shifting of the cone-center is easily carried out; in fact, it is difficult to avoid it unless some definite means of preventing it is adopted. Fig. 1 shows very clearly the structure of a cone with a "drawn" center, and in this instance the effect was entirely unintentional.

The irregularities in the results obtained by Cornish sampling, together with the cost of operation and the amount of room required, soon brought about what is known as "split-shovel" sampling, in which the ore is thrown from a broad shovel, handled by one operator, upon a narrow "U"-shaped shovel, held by another workman, usually directly over a car or wheelbarrow, as shown in Fig. 6. This method, while it requires two men to do what normally appeared to be the work of one, was cheaper than Cornish quartering, but it proved no great improvement over the latter in point of accuracy, since carelessness in almost any direction interferes seriously with the results.

The earliest attempts at mechanical sampling were made by subdividing a falling stream of ore; a process based on the supposition that an ore-stream could be mixed so as to be perfectly homogeneous. Both analysis and experience have shown that this ideal condition is impossible, and mechanical devices for taking a portion of the ore-stream all of the time have been almost entirely displaced by machines designed to take all of the ore-stream for a portion of the time. It is not practicable to produce a stream of ore which shall be continuous in value through every part of its length any more than it is possible to produce a stream of ore that is constant in value throughout its width; but by taking a small sample entirely across a falling-stream at very short intervals it is found that, while no single cut would give an exact representation of the composition of the entire lot, the average of thousands of these small samples

FIG. 1.—SAMPLE BISECTED BY A SHEET OF GLASS, SHOWING PROPORTION OF COARSE AND FINES.

FIG. 2.—SAMPLE SPREAD OUT INTO A PANCAKE.

FIG. 3.—SAMPLE CONED AROUND A ROD.

FIG. 4.—SAMPLE CONE WITH DRAWN CENTER.

FIG. 5.—SAMPLE WITH DRAWN CENTER SPREAD OUT.

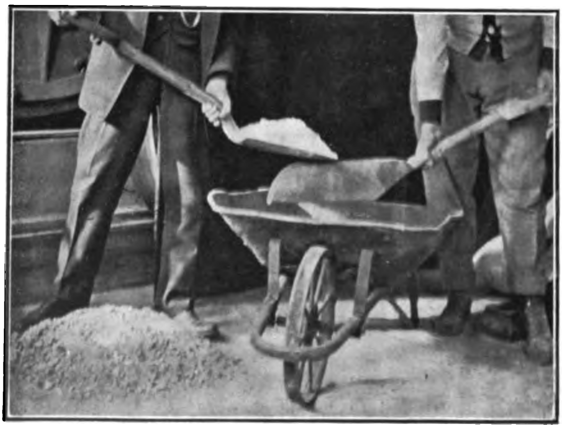

FIG. 6.—U-SHOVEL SAMPLING.

is so nearly correct that results can be duplicated within very narrow margins, or, in other words, that individual errors are balanced. This was not the case with the devices used for taking a portion of the stream all the time, since the errors due to feed, inclination of spouts, or wear on the bottoms of the spouts are constant, and do not vary during the time the samples are being taken.

Almost coincidental with the discovery of the fact that accurate samples could be obtained by taking all of the stream for a portion of the time, came a very considerable improvement in rock-crushing machinery, so that the modern engineer has a much better opportunity to construct a satisfactory plant than the builder had 20 or 30 years ago. Not only are the rock-breaker and rolls of to-day greatly improved in design, but the manufacturers have availed themselves of modern cheap steel to give all parts an excess of strength over any possible strain, while the use of alloy-steels for the wearing-surfaces permits the machines to be kept in much better repair, and requires fewer stoppages for renewals. For sampling-work, crushers and rolls can now be had which are almost as well made as the ordinary steam-engine, and so designed as to give complete accessibility for renewals and for cleaning.

Gyratory breakers of the Gates type have the advantage of delivering a very uniform product, and in crushing ores that are hard and dry this type forms by all odds the best initial crushing-machine; but with ores that carry wet clay, slate, or other substances which will "pack," it is necessary to use a swinging-jaw crusher, preferably of the Blake type. Rock-breakers may be set to crush to any desired fineness, but it has been found that too great a reduction in the size of the product very materially reduces the capacity. In large crushers it is not usually advantageous to attempt to crush below 2 in. in size.

First-class rolls are now always belt-driven, which eliminates the noise and danger attending the operation of the old-fashioned trains of gears. The best practice in roll-crushing is to crush not smaller than half the diameter of the particles fed to any given machine. This rule gives approximately the maximum crushing-capacity with the minimum production of fines and the lowest expenditure for power and metal. Rolls

require a steady feed, and one which is uniform across the entire width of the shell; consequently, nearly all modern rolls are equipped with some feeding-device. In sampling-mills the shaking-tray is generally used on account of the ease with which the rate of feed can be inspected, and the great facility with which such feeders can be cleaned after each lot of ore has been run.

For fine-grinding machines, the coffee-mill type still successfully holds its own against most of the newer devices, although the modern sample-grinder is much heavier, better built, and more easily cleaned than its predecessors.

The first mechanical samplers were imitations of Cornish quartering, the "whistle-pipe" being the most common type. With ore finely crushed, fairly dry, well mixed, and entirely free from strings and rags, and with the dividers new and exactly centering the pipe, fairly good results could be obtained by this method; but as these conditions never existed in practice, and as the edges of the cutters wore rapidly, thereby moving the dividing-line back from the center, this form of sampling-machine was soon discarded, and I believe has now fallen into absolute disuse.

Following the whistle-pipe sampler came the various forms of mechanical split-shovels; but as there was no place in a spout, no matter how wide or carefully built, where a single "U"-shaped spout could be placed to take a sample which would represent the entire width of the stream, this form also was soon discarded.

More recently this splitter has been revived by an adaptation of the ordinary hand-operated splitter (see Fig. 12), in which numerous small spouts are so arranged across the entire width of a larger one that the main ore-stream is divided into a great number of smaller ones, the even numbers being deflected to the right and the odd numbers to the left. This plan works very well on the first division, but as it effects a reduction of only 50 per cent. in the volume, the operation must be carried further, and the streamlets forming the sample centered into a broad stream, which, in turn, passes over another set of splitters, the operation being repeated as often as necessary to reduce the sample to the desired size. It has been found, however, that the mixture of the streamlets after their union is far

from perfect, and that there is a considerable difference in the amounts of coarse and fines taken by the sample side of the second cutter, depending on its position relative to the cutter above. If the sample-compartments in the second cutter are directly below the sample-compartments in the overlying splitter, they receive the centers of the streamlets, while the "spread" passes into the reject, and the sample at each step in the bank of cutters receives an amount of fines slightly in excess of the average, thereby seriously affecting the value of the sample, provided there is, as is usually the case, a difference between the metallic contents of the coarse and the fines. Conversely, when each cutter in the bank is placed so as to take the "spread" from the cutter above it, the sample will have less than its due proportion of fines. This disadvantage could be obviated by placing a shaking-tray between each set of dividers, or perhaps even better, by moving one divider horizontally across the other, so that each set of cutters would take all parts of the streams from the cutters above them. This arrangement, however, would require considerable head-room, and give a machine which would make a large amount of dust—a feature which is always objectionable in a sample-room.

The latest types of samplers are designed to overcome the difficulties just described, and are usually known as "time-sampling machines," from the fact that they deflect the entire stream into the sample-compartment for a varying portion of the time, depending on the percentage of sample required. Treating the falling stream of ore as a ribbon, they cut sample-sections directly across its entire width, these portions varying in shape and size with the mechanism employed. Of the many types that have been invented and patented, only three have come into general use, and Fig. 7 shows the shapes of the sample-sections taken by these three machines.

A represents a sample cut from the falling stream of ore by the Charles Snyder 20-per cent. sampler, with four radial intake-spouts, making 7.5 rev. (or 30 samples) per minute; delivery-spout 5 by 25 in. (This sampler is not to be confounded with the Snyder sampler.) It will be seen, on this machine, that an attempt has been made to combine the old-fashioned continuous sample with the time-sampling system by arranging

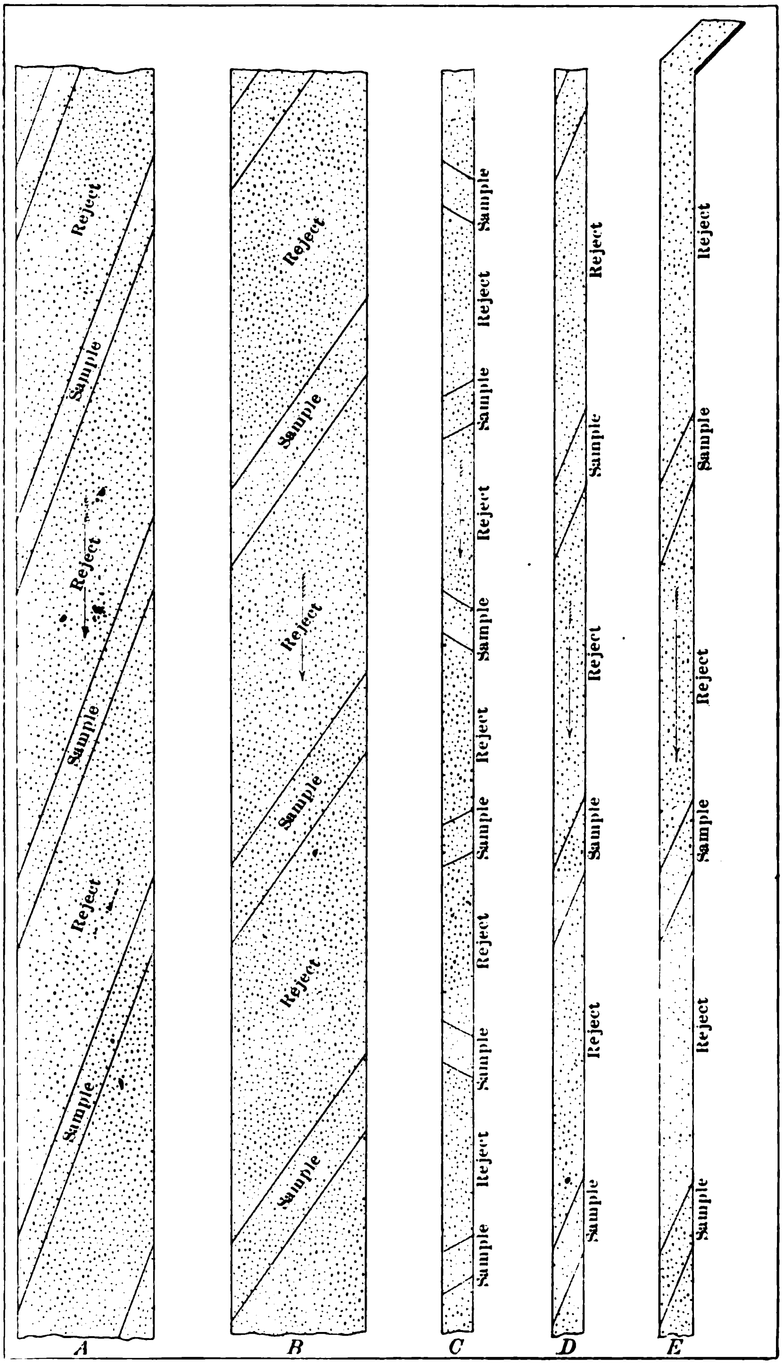

Fig. 7.—Shapes of Sample-Sections Taken by the Charles Snyder, Brunton, and Vezin Samplers.

the delivery-pipe and intake-spouts so that as one intake-spout passes out of the stream another enters it on the opposite side.

B represents a Charles Snyder 20-per cent. sampler, with two radial intake-spouts, taking 15 samples per minute; delivery-spout 5 by 25 in. This machine does not take a continuous sample, but has the advantage that the intake-spouts, for a given percentage of sample, have double the width, and are therefore much less liable to throttle or choke; at the same time there is no reason why the sample should not be as accurate as that taken with the other type of Charles Snyder sampler.

C represents the sample taken by the Brunton 20-per cent. sampler, taking 54 samples per minute; delivery-spout 5.75 by 5.75 in., cutting-edges parallel.

D represents the sample from a Vezin 20-per cent. sampler with two radial intake-spouts, taking 30 samples per minute; delivery-spout 6 by 6 inches.

E shows the sample taken by a modified form of sector-sampler, which, often through accident and sometimes by design, has come into too-general use.

Both the Charles Snyder and the Vezin samplers have sector intake-spouts revolving on a vertical axis, the only difference between the machines being that the delivery-spout in the Snyder sampler is an annular quadrant, *D*, in Fig. 8, while the Vezin delivery-pipe is either square or rectangular, *E*, in Fig. 9. In order to take a correct sample the cutting-edges of the sector intake-spouts on both of these machines must be exactly radial, as shown in Figs. 8 and 9, otherwise they will include more degrees of arc at one part than at another; and consequently the percentage of sample taken from all parts of the delivery-pipe will not be the same, as is shown by Fig. 10, in which the cutting-edges are not radial to the center of rotation. This, while by no means an exaggerated example of this form of distortion, shows a 74/360, or 20.8-per cent., sample taken on one side of the ore-stream and 88/360, or 24.4 per cent., on the other. If the falling stream of ore were perfectly homogeneous this arrangement would not make any difference, but it is well known that the ore-stream is not uniform, especially in an inclined spout, in which the coarse, rapidly-moving particles go bounding along on the top, while the finer portions

hug the bottom, and on leaving the spout the coarse is projected a considerable distance and the fines drop almost vertically, which gives a sorted falling stream, with coarse on one side and fines on the other. With a tangential feed to a sector intake this sorting-action does not seriously affect the sample if the delivery-spout is perfectly level and free from ridges which would deflect the particles across the stream; but with a radial feed used, as shown in Fig. 9, and the intake sample-

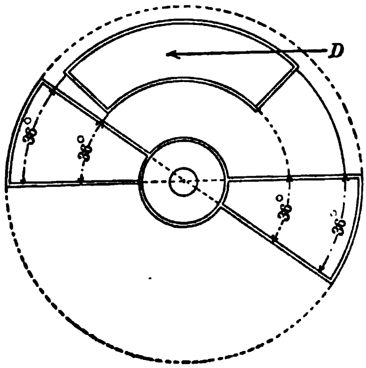

FIG. 8.—DELIVERY-SPOUT OF CHARLES SNYDER SAMPLER. CUTTING-EDGES RADIAL.

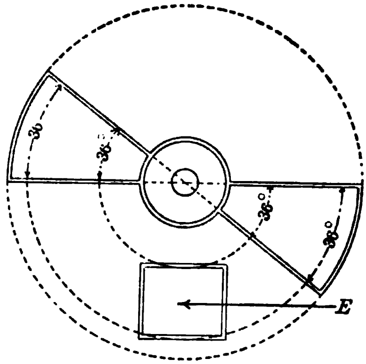

FIG. 9.—DELIVERY-SPOUT OF VEZIN SAMPLER. CUTTING-EDGES RADIAL.

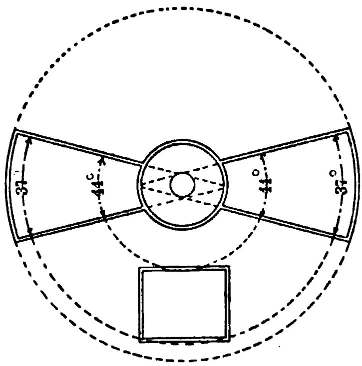

FIG. 10.—CUTTING-EDGES NOT RADIAL.

spout edges not radial, as shown in Fig. 10, it will readily be seen that a larger proportion of coarse than of fines is taken into the sample.

Since the cutting-edges of this class of samplers, Fig. 11, are necessarily maintained in a horizontal position, they are very liable to become overhung with strings, burnt fuse, and drill-rags, which the mill attendants often endeavor to remove

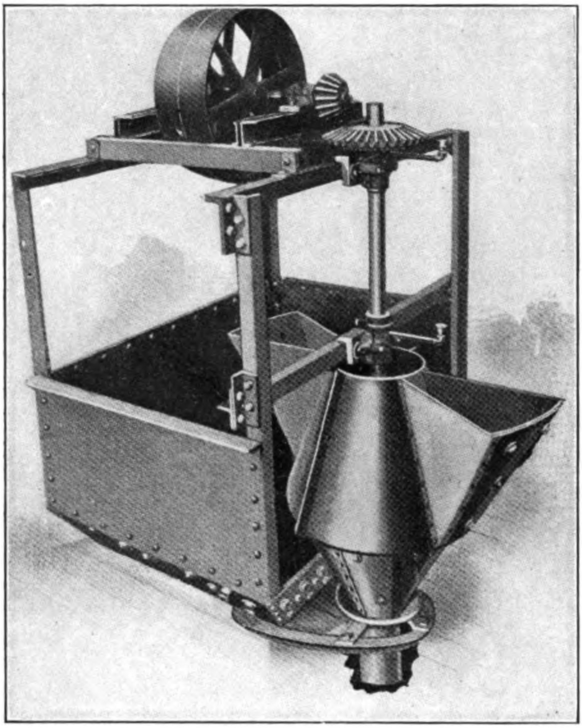

FIG. 11.—VEZIN SAMPLER, SHOWING HORIZONTAL CUTTING-EDGES.

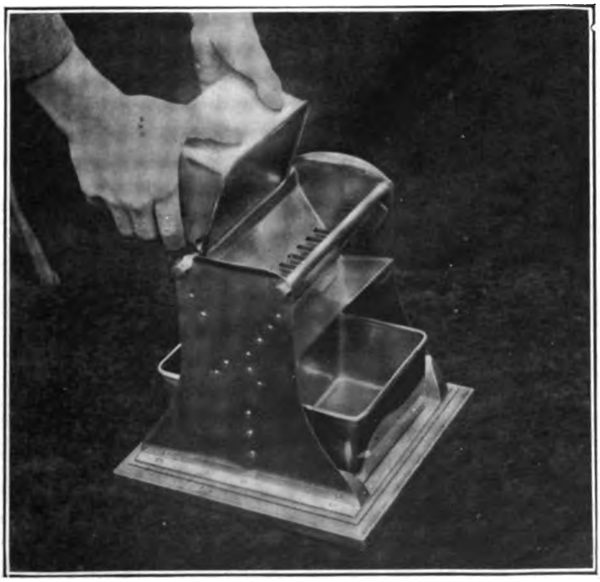

FIG. 12.—TAYLOR & BRUNTON SPLITTER.

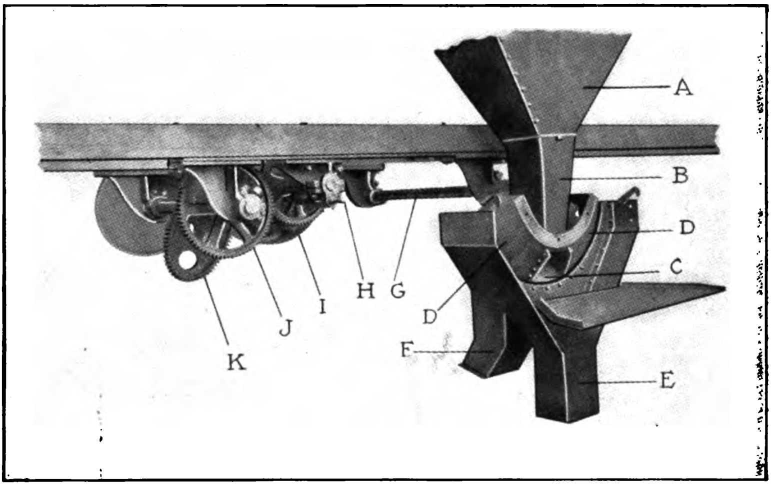

FIG. 13.—THE BRUNTON TIME-SAMPLER. FRONT VIEW.

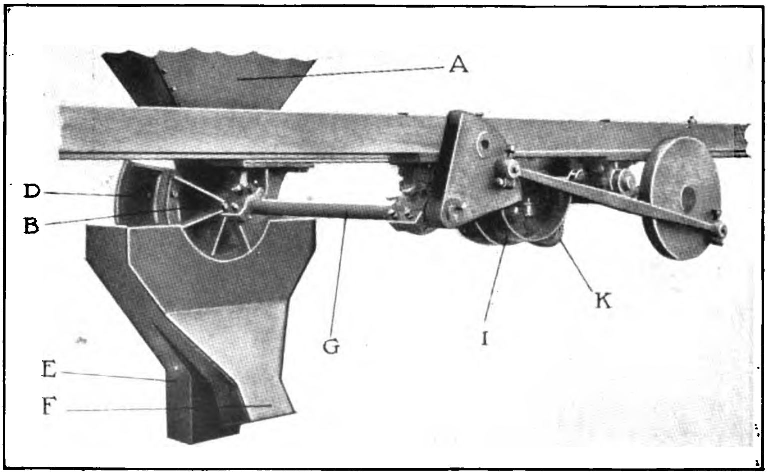

FIG. 14.—THE BRUNTON TIME-SAMPLER. REAR VIEW.

by pounding the sides of the spout while the machine is in motion, thereby distorting the form of the intake-spout very considerably from a true sector, and rendering it impossible to obtain a correct sample unless the delivery-stream is perfectly homogeneous, which is never the case. The great length of the radial edges of the sector intake-spout renders them, of course, peculiarly susceptible to be thrown out of alignment, and manufacturers of this class of machines should do something to shorten up the length of the radial edges, or stiffen them to prevent accidental distortion. At first sight it might be thought that this could be accomplished by reducing the size of the sector, but experience has shown that the width of any spout, delivery, or intake should be something more than three times the greatest diameter of the coarsest particle passing through it; otherwise, a bridging effect occurs which affects the flow and often chokes up the spout. It is, therefore, good practice to make the width of the feed- and intake-spouts four times the diameter of the coarsest particles passing through them.

The Brunton time-sampler oscillates in a vertical plane through an arc of 120° instead of revolving in a horizontal plane like the sector-intake machines, an arrangement which permits the use of a rectangular intake-spout with cutting-edges so short that accidental distortion is impossible, while the tilting of the cutters at the end of the swing materially assists in dislodging any rags or strings which may have fallen on the cutting-edges. This construction requires less head-room than any other system, which effects a great saving in the cost of mill-construction, since it not only reduces the necessary height of the building but shortens all spouts and conveyors. The design of this machine cannot be very clearly shown in a linear drawing, but may be readily understood from Fig. 13, which is a front view of the sampler, having the housing open for cleaning, and Fig. 14, which is a rear view. The various parts are explained as follows: *A*, receiving-hopper from crusher or rolls; *B*, delivery-spout; *C*, sample-intake; *DD*, "reject" divisions; *E*, housing-spout leading to the sample-bin; *F*, reject-spout leading to the shipping-bins; *G*, oscillator-shaft; *H*, gear-shift; *I*, driving-pulley; *J*, spur-gear; *K*, eccentric gear. Ordinarily the machine is driven by

the spur-gear, *J*, in which case a 20-per cent. sample is taken, but when a 5-per cent. sample is required the gear is slipped along the shaft, disengaging the spur and bringing the eccentric gear, *K*, into play. Another advantage in the use of this machine is that, as the discharge of the ore from the sampler is assisted by centrifugal force instead of being retarded thereby, as is the case with all sector machines, it can be run at a much higher rate of speed, thereby increasing the number of samples per minute. This arrangement insures greater accuracy, since the more samples which can be cut from the falling ribbon without "batting" the ore too vigorously with the sides of the cutters, the better are the chances for obtaining an exact average of the stream. A study of the relations between oscillator, rocker-arm, and disk-crank, Fig. 14, will show how this device takes a comparatively small sample with a large intake-spout.

While there seems to be a general impression among mining-men that high-grade ores are more difficult to sample correctly than those of low grade, there is no reason for this assumption. The difficulty of sampling accurately increases directly as the difference between the value of the highest- and the lowest-grade material contained in the lot, and is at its maximum when the values are carried in large masses of metallics or crystals of very rich minerals occurring in barren rock.

If we imagine a lot, for instance, of Cripple Creek ore, composed entirely of barren gangue and one solitary piece of calaverite, it would be manifestly impossible to sample such a lot of ore without crushing, since in any subdivision either the sample or the reject would contain all of the mineral.

Suppose this lot to be subjected to a slight crushing and the solitary piece of mineral broken into three fragments, then dividing the lot into halves would at the best throw 50 per cent. more value into the one half than into the other; it is therefore clearly manifest that in order to obtain a sample which shall correctly represent this or any other lot, it is necessary to crush it to such a degree of fineness that one particle more or less taken into the sample shall not materially affect its metallic content. In other words, the maximum error is determined by the ratio of the weight of the largest particle of metal or high-grade mineral to the weight of the entire lot. At this point another condition must be considered. In any lot

of ore it is easy to see that the chances of finding a full-sized piece of the highest-grade material would be much greater on

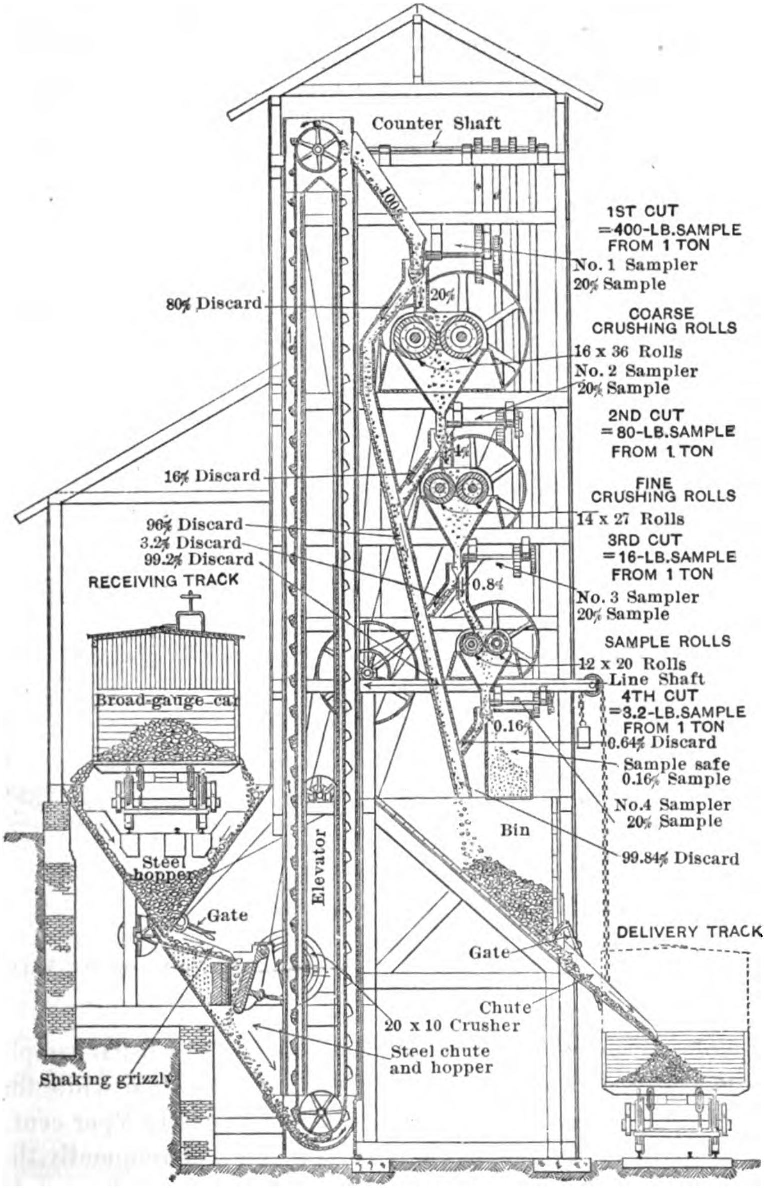

Fig. 15.—Taylor & Brunton Sampling-System.

a lot of ore crushed to 0.25-in. cubes than in a lot crushed in 1-in. cubes, therefore accuracy demands that the ratio between

the weight of the largest particle and the entire lot shall increase directly as the fineness.

In this particular, practice and theory are in complete accord, and all of the latest and most improved mills practice alternate crushing and subdivision from the coarsest size down to the finest. It is customary at each successive stage to reduce the diameter of the coarsest particles one-half, thus decreasing

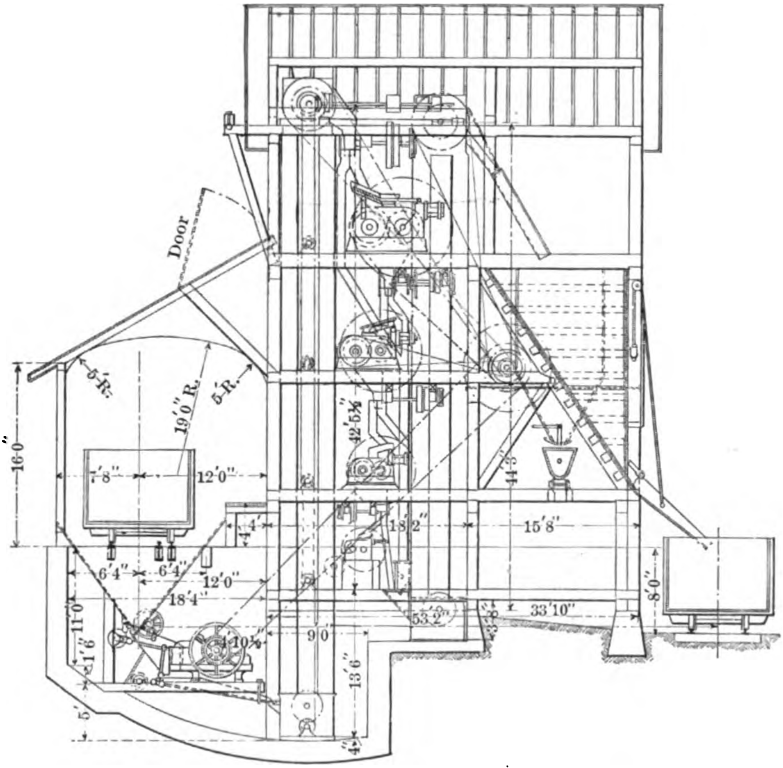

FIG. 16.—VERTICAL SECTION OF THE TAYLOR & BRUNTON SAMPLING-MILL, SILVER CITY, UTAH.

the volume to one-eighth, or 12.5 per cent. The usual sample taken at each successive stage is 20 per cent., so that while the size of the particle at each step has been reduced 12.5 per cent., the amount of sample taken is 20 per cent., consequently the ratio between the weight of the largest particle and the weight of the sample rises steadily from the beginning of the series of operations to the end, thereby meeting the conditions theoretically necessary to an accurate determination of value.

An ideal sampling-mill, where the situation and nature of the service will permit this form of construction, is shown in Fig. 15. This plant is entirely automatic, and when the ore is received in hopper-bottom cars no manual handling is required at any stage, while the sample is automatically delivered into a

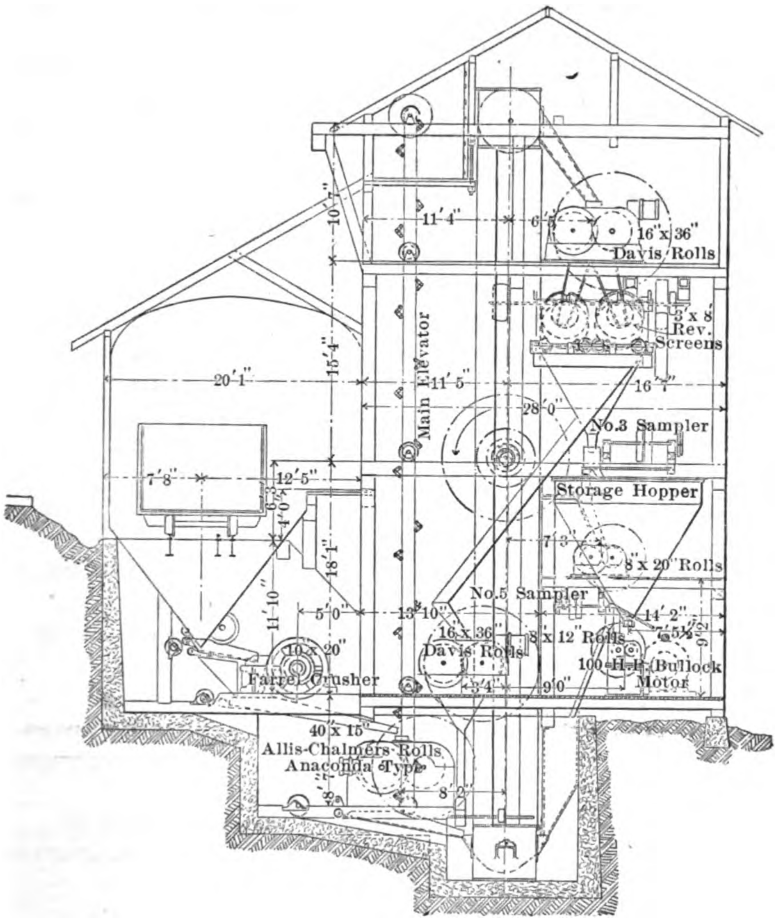

FIG. 17.—VERTICAL SECTION OF THE MATTE AND SULPHIDE SAMPLING-MILL OF THE TINTIC SMELTING CO., SILVER CITY, UTAH.

locked steel safe. To simplify the drawing, the roll-feeders have been omitted.

Fig. 16 is a vertical longitudinal section of the new Taylor & Brunton mill at Silver City, Utah, completed January, 1909. Like the plant shown in Fig. 15, it is automatic throughout,

electric driven, and contains every modern device for facilitating crushing, sampling, and cleaning, the latter operation being performed by compressed air.

TAYLOR & BRUNTON SAMPLING WORKS, SILVER CITY, UTAH.

Calculations based on 25 ton lot. Capacity 60 tons per hour

3 rail receiving tracks R.G.W., San Pedro and Eureka Hill Rys.

Steel ore hopper under railway tracks 14' 6" wide x 15' 0" long.

Shaking grizzly separating coarse and fines.

10" x 20" Farrel-Bacon crusher crushing to 2-1/2" cubes. 260 RPM.

Shaking tray elevator feeder 1-1/4" stroke. 250 RPM.

55' 0" vertical elevator belt 20", buckets 6" x 18". Speed 375 RPM.

No.2 Brunton 20% Sampler 7" C-C 19 RPM.

Reject 80% — 20% Sample = 10,000 lbs. = 1/3500-

Shaking tray roll feeder 3/4" stroke. 233 RPM.

16" x 36" Davis belted rolls crushing to 1" cubes. 50 RPM.

No.3 Brunton 20% Sampler 5-3/4" C-C. 30 RPM.

Reject 16% — 4% Sample= 2000 lbs.= 1/11,000.

Shaking tray roll feeder 5/8" stroke. 235 RPM.

15" x 27" Gates belted rolls crushing to 3/8" cubes. 68 RPM.

No.4 Brunton 20% Sampler 4-1/2" C-C. 26 RPM.

Reject 3.2% — 0.8% Sample= 400 lbs.= 1/43,000.

Shaking tray roll feeder 1/2" stroke. 180 RPM.

8" x 20" Davis belted rolls crushing to 1/8" cubes. 80 RPM.

No.5 Brunton 20% Sampler 3-1/2" C-C- 33 RPM.

Reject 0.64% — 0.16% Sample= 80 lbs.= 1/240,000.

Shaking tray roll feeder 1/2" stroke. 185 RPM.

8" x 12" Davis finishing rolls crushing to 14 mesh. 100 RPM.

Locked steel sample safe.

Covered steel sample buggy.

Locked cutting room with steel floor and observation windows.

T.& B. precision 3/4" div. splitter to 10-12 lbs.=1/1,100,000.

Electric drier. Temperature 250° F.

Two Engelbach grinders to 50 mesh cone makers, 86 RPM.

1/2" division T.& B. precision splitter to 20-24 ozs.=1/7,000,000.

Bucking Board to 120 mesh.

Rubber rolling cloth five minutes.

1/4" div. T.& B. precision splitter to 4 sample sacks 5-6 ozs.each.

Loading out Bins.

Reject Elevator.

FIG. 18.—FLOW-SHEET, TAYLOR & BRUNTON SAMPLING WORKS, SILVER CITY, UTAH.

A good example of a modern crushing, screening, and sampling plant is shown in Fig. 17, which is a longitudinal section through the new matte and sulphide mill of the Tintic Smelting Co. at Silver City, Utah.

It is not the purpose of this paper, however, to take up and illustrate details of construction, but rather to show the methods which are being employed to produce the best results in the

SYNOPSIS OF MACHINERY AND METHODS.

MATTE & SULPHIDE MILL, TINTIC SMELTING COMPANY:

Calculations based on 35 ton lot. Capacity 30 tons per hour.

3 rail tracks, R.G.W., San Pedro and Eureka Hill Rys.

Steel receiving hopper under railway tracks, 14' 6" wide x 15'0" long.

Shaking grizzly separating coarse and fines.

10" x 20" Farrel-Bacon crusher, 260 RPM, crushing to 2" cubes.

Shaking tray roll feeder 1-1/4" stroke, 250 R.P.M.

15"x 40" Allis-Chalmers, Anaconda Type, belted rolls, 45 RPM, crushing to 1" cubes.

Shaking tray elevator feeder 1-1/4" stroke. 250 R.P.M.

55' 9" C-C elevator belt 20", buckets 6"x 18". Speed 355' P.M.

16" x 36" Davis belted rolls, 50 RPM. , crushing to 1/2" cubes.

3 revolving screens each 30" dia. 8'0" long, 2-1/2 mesh, 1/4" openings.

Sized Product — Oversize.

No.3 Brunton 2% Sampler 15 RPM 3-5/16" C-C of cutting edges.

16" x 36" Davis finishing rolls, 50 RPM, 1/4" cubes.

Reject 98%

2% Sample

Elevator

Storage hopper

= 1000 lbs.=1/350,000

Screens

Robbins belt

Sulphide beds

Shaking tray transfer feeder 1/2" stroke. 180 RPM.

8" x 20" Davis belted sample rolls crushing to 5 mesh. 75 RPM.

No.5 Brunton 10% Sampler, 31 RPM, 1-3/4" C-C.

Reject 1.8% — 0.2% Sample=100 lbs.= 1/440,000.

Auxiliary hopper and run to

storage hopper during cleanup

Shaking tray roll feeder 1/2" stroke. 185 RPM.

8" x 12" Davis finishing rolls crushing to 14 mesh. 100 RPM.

Locked steel sample safe

Covered sample buggy.

Locked cutting room with steel floor and observation windows.

T.& B. precision splitter to 10-12 lbs.= 1/1,100,000.

Covered and locked steel bucket to grinding room in sampling mill.

FIG. 19.—FLOW-SHEET, TINTIC SMELTING CO., SILVER CITY, UTAH.

valuation of ores; an operation which means everything to the mining and metallurgical industries. One of the first requisites for successful mining is an accurate knowledge of what a

property is producing, and this of necessity involves correct sampling, both underground and on the surface. The first

SYNOPSIS OF MACHINERY & SAMPLING SYSTEM

AT THE WESTERN ORE PURCHASING CO'S. WORKS, M I L L E R S, NEVADA.

Computations based on 25 ton lot. Capacity 70 tons per hour.

On main line of Tonopah & Goldfield Railroad Co.

Four storage tracks. 6000 feet private track.

Receiving Hopper under railroad track.

Power 75 H.P. Westinghouse Motor. 710 R.P.M. 2200 Volts

Two No.4 Gates crushers Style K, crushing to 2" 420 RPM

Elevator 71' 6" C-C. Buckets 16" x 6" x 6". Speed 420' P.M.

No.1 Special 20% C. Snyder Sampler, delivery spout 7"x35" 10 RPM.

Discard 80% 20% Sample = 10,000 lbs. = 1/7,000.

16" x 36" Davis belt-driven rolls, crushing to 1". 70 RPM.

Elevator 31' 3" C-C. Buckets 4" x 8". Speed 340' P.M.

No.1 Chas. Snyder 20% Sampler, delivery spout 5"x 25". 10 RPM.

Discard 16% 4% Sample = 2000 lbs. = 1/11,000.

14" x 30" Davis belt-driven rolls, crushing to 3/8". 110 RPM.

Elevator 28' 9" C-C. Buckets 3" x 5". Speed 340' P.M.

No.2 Chas. Snyder 20% Sampler, delivery spout 3"x 15". 10 RPM.

Discard 3.2% 0.8% Sample = 400 lbs. = 1/42,000.

12" x 20" Davis belt-driven rolls, crushing to 6 mesh. 110 RPM.

No.3 Chas. Snyder 20% Sampler, delivery spout 2" x 10" 10 RPM.

Discard 0.64% 0.16% Sample = 80 lbs. = 1/670,000.

Sample Hopper car, bottom discharge.

Enclosed Sample Room. 10 H.P. Motor 710 R.P.M.

7" x 14" Stearns-Roger rolls, crushing to 8 mesh. 150 RPM.

3/4" div. riffle box to 50-150 lbs. = 1/1,040,000.

Regrind through closed 7" x 14" rolls, to 16 mesh.

Grinding Room.

1/2" div. riffle box to 12-15 lbs. = 1/1,600,000;

Electric drier. Temperature 240°- 380° F.

Two No.2 Englebach Grinders to 50 mesh. Speed 60-130 RPM.

Rubber rolling cloth three minutes.

1/2" div. riffle box to 20-25 ounces = 1/7,000,000.

Grinder and Bucking Board to 120 mesh.

Rubber rolling cloth 10 minutes.

1/4" div. riffle to 4 sample sacks 5.5 ounces each.

Loading Hopper

R.R. Cars standing on 50' 100 ton Fairbanks reg. beam scales.

Reject Elevator.

FIG. 20.—FLOW-SHEET, WESTERN ORE PURCHASING CO., MILLERS, NEV.

essential to success in all metallurgical work is a knowledge of the exact value of the ore going into the works, and of the different products issuing therefrom.

In order to show the methods of operation in vogue in different districts, I present Figs. 18, 19, 20, and 21, which contain the flow-sheets of a number of the newest and largest sam-

SYNOPSIS OF MACHINERY AND METHODS AT

THE AMERICAN S. & R. COMPANY SAMPLER No. 2, MURRAY, UTAH.

Computations based on 25 ton lot. Capacity 50 tons per hour.

Rio Grande Western & S. P. L. A. & S. L. Railways.

Stub end elevated receiving track 10' above mill floor.

Wide chute at level car floor, feeding direct to crusher.

Gates crusher #4 style D, crushing to 3".

Ninety foot elevator, buckets 6" x 8" x 12".

Vezin 20% 20" radius double spout sampler. 13 R.P.M.

Reject 80% — 20% sample = 10,000 lb. = 1/2000.

Gates crusher #2 style H, crushing to 1-1/4".

Vezin 20% 20" radius sampler.

Reject 16% — 4% sample = 2,000 lb. = 1/5000.

Loading bins over railroad track.

15" x 26" Gates belted rolls, crushing to 3/4".

70 foot elevator 4" x 4" x 6" buckets.

Vezin double spout 20% sampler.

Reject 3.2% — 0.8% sample = 400 lb. = 1/6300.

12" x 20" high-speed Allis Chalmers rolls, crushing to 3/8".

Vezin double spout 20% sampler.

Reject 0.64% — 0.16% sample. = 80 lb. = 1/8500.

Steel safe in grinding room.

Entire sample carried over to #3 mill.

Run through 12" x 20" rolls, crushing to 3/16".

3/4" Rhodes splitter to 75 lb. = 1/62500.

Surface dryer in #4 mill grinding room.

Two Engleback grinders to 50 mesh.

Auxiliary ore bin.

1/4" riffle to 48-50 ounces = 1/16,000,000.

Bucking board to 100 mesh Ag. and 120 mesh Au.

Gold pan mixing five minutes.

1/4" riffle to four 8 oz. pulp samples.

FIG. 21.—FLOW-SHEET, AMERICAN SMELTING & REFINING CO., SAMPLER NO. 2, MURRAY, UTAH.

pling-works, clearly showing every detail of the process, and the machinery employed in the alternate operations of crushing and subdivision, as well as the increase of ratio as the final stages are reached. This style of flow-sheet was originally

type-written on ordinary 8.5 by 13 in. paper, perforated for a loose-leaf binder. In this way flow-sheets of many classes of operations may be preserved in convenient form.

These flow-sheets show considerable differences at all stages, and a great divergency in the methods of subdividing the final sample. Too many manual operations are in use, and there is no doubt that the complete elimination of the personal equation by using a small Taylor & Brunton splitter with ⅜-in. riffles (shown in Fig. 12) gives by far the most accurate subdivision.

To show how closely results between different mills and repeat-sampling in individual mills may be made to check, the following examples, taken at random, should suffice:

Table I.—*Sampling-Results, Taylor & Brunton Sampling Co., Cripple Creek, Colo.*

Lot No.	Sample.	Resample.
	Gold.	Gold.
	Ounces per Ton.	Ounces per Ton.
3192	3.62	3.64
3198	5.04	5.015
3219	2.70	2.67
3235	3.18	3.16
3310	1.17	1.17
3324	6.52	6.51
3340	0.71	0.78
3388	1.70	1.84
3424	9.24	9.20
3471	30.64	30.52

Table II.—*Sampling-Results, Taylor & Brunton Sampling Co., Cripple Creek, Colo.*

Lot No.	Mine.	First Sample.		Resample.		Settlement.
		Gold.		Gold.		
		Mill-Assay.	Mine-Assay.	Mill-Assay.	Mine-Assay.	
		Oz. per Ton.	Oz. per Ton.	Oz. per Ton.	Oz. per Ton.	Oz. per Ton.
4514	Sacramento..........	2.22	2.24	2.22	2.23	2.225
4604	Little Clara.........	115.05	115 25	114.90	115.20	115.03
4705	Mary Cashen.........	1.11	1.10	1.07	1.09	1.08
4726	Midget.................	1.27	1.30	1.30	1.35	1.325
4853	Independence, Ltd.	1.36	1.35	1 29	1.30	1.295
4914	Bon. King..........	0.53	0.55	0.55	0.56	0.555
5062	Little Clara..........	1.77	1.72	1.75	1.74	1.745
5272	Old Abe...............	1.27	1.24	1.27	1.28	1.27
5753	Independence, Ltd.	2.33	2.34	2.34	2.36	2.35
5913	Little Clara..........	12.62	12.58	12.69	12.68	12.695

TABLE III.—*Sampling-Results, Taylor & Brunton Sampling Co., Cripple Creek, Colo.*

Lot No. of Mixture.	Original Purchase.		Mixture.	
	Weight.	Gold-Assay.	Mathematical Average.	Mechanical Sample.
	Pounds.	Ounces per Ton.	Ounces per Ton.	Ounces per Ton.
5394	17,588 9,646 11,348	0.98 1.17 0.875	0.996	1.00
5496	17,405 6,615 17,123	0.98 0.895 0.995	0.972	0.975
5799	422 12,851 175 21,278	8.24 2.225 8.50 1.85	2.099	2.14
5890	19,090 8,761 8,852	1.925 1.97 1.89	1.927	1.93
3465	5,274 17,935	2.10 1.89	1.937	1.97
3678	3,795 17,122 11,357 6,592	1.88 1.49 1.345 1.465	1.481	1.52
3850	3,633 16,803 8,360 11,222 3,731	3.365 4.675 5.82 3.73 36.445	7.252	7.24
4170	18,605 18,621 11,937 8,593	0.83 0.77 1.42 0.98	0.954	0.92
4292	17,848 15,435 17,436	1.165 0.615 1.12	0.982	0.96
4319	4,014 15,611 13,334 11,712	2.835 2.24 3.35 2.58	2.71	2.75

TABLE IV.—*Sampling-Results, American Smelting & Refining Co., No. 2 Sampling-Mill, Utah, Using Vezin Samplers.*

Number.	Size of Lots. Tons Dry.	First Sample.		Resample.	
		Gold.	Silver.	Gold.	Silver.
		Oz. per Ton.	Oz. per Ton.	Oz. per Ton.	Oz. per Ton.
1	131	5.18	1.1	5.02	1.1
2	138	4.67	trace	4.82	trace
3	85	2.45	1.0	2.45	1.0
4	75	3.49	5.3	3.45	5.5
5	104	2.48	1 0	2.41	1.0
6	138	2.31	trace	2.39	trace
7	97	2.43	2.0	2.31	2.0
8	96	2.43	1.4	2.38	1.2
9	83	2.47	1.5	2.48	1.7
10	91	5.08	trace	4.94	trace
Average.....	103.8	3.299	1.33	3.265	1.35

TABLE V.—*Sampling-Results, Western Ore Purchasing Co. Plants.*

Using Charles Snyder Samplers.

Miller's Plant:

	First Sample.		Resample.	
	Gold. Ounces Per Ton.	Silver. Ounces Per Ton.	Gold. Ounces Per Ton.	Silver. Ounces Per Ton.
Lot No. 4979, Assayer A, ...	0.21	36.45	0.21	36.35
Assayer B, ...	0.20	36.35	0.20	36.85
Average, ...	0.205	36.40	0.205	36.60

Columbia Plant:

	First Sample. Gold. Ounces Per Ton.	Resample. Gold. Ounces Per Ton.
Lot No. 844, average of two assays, ...	5.393	5.37

Hazen Plant:

	First Sample.		Resample.	
	Gold. Ounces Per Ton.	Silver. Ounces Per Ton.	Gold. Ounces Per Ton.	Silver. Ounces Per Ton.
Lot No. 1131, ...	1.76	4.50	1.743	4.65

TABLE VI.—*Sampling-Results, Columbia Plant.*

Lot Mixture No. 473.

Lot Number.	Dry Weight.	Assay Gold.	Assay Silver.	Gold-Content.	Silver-Content.
	Pounds.	Oz. per Ton.	Oz. per Ton.	Ounces.	Ounces.
972	78,884	1.91	1.10	75.33	43.38
961	78,408	1.82	0.90	71.35	35.28
974	78,837	1.69	0.80	66.62	31.53
979	37,352	4.23	......	79.00	....
1145	7,119	0.30	161.40	1.07	574.50
	280,600	....	....	293.47	684.69
Mathematical average.....		2.09	4.89		
Actual sample of mixture:					
	280,364	2.07	4.83	290.17	676.29

Table VII. gives a comparison on a lot of Bullfrog Pioneer ore sampled at Columbia plant, and afterwards screened through a ⅜-in. screen at Hazen; fines sold to reverberatory and coarse to blast-furnace smelters, actual weights and moistures having been determined both on the fines and the coarse, which makes a showing of a slight loss in weights.

TABLE VII.—*Sampling-Results, Columbia Plant.*

Lot No. 1017.	Dry Weight.	Assay Gold.	Total Gold-Content.
	Pounds.	Ounces per Ton.	Ounces.
	122,189	3.71	226.66
		After screening:	
Fines...........	36,909	6.06	111.83
Coarse.........	84,760	2.75	116.55
	121,669		228.38

Table VIII. gives a comparison of assays and total ounces of gold contained in four lots of Engineers' Lease ore from the property of the Florence-Goldfield Mining Co., in Goldfield, Nev., sampled at Columbia plant and afterwards screened through ⅜-in. punched screen at Hazen plant, and the coarse and fines sampled separately after screening.

The dry weights show the same in each case, due to the fact that the fines after screening at Hazen were actually weighed and moistured, thus determining the exact dry weight, which was deducted from the total purchased dry weight, making a figured dry weight of the coarse.

Table VIII.—*Sampling-Results, Hazen Plant.*

Lot No.	Dry Weight of Ore.	Assay Gold.	Total Gold-Content.
	Pounds.	Ounces per Ton.	Ounces.
861	70,636	7.26	256.41
872	72,682	7.45	270.74
	143,318		527.15
		After screening:	
Fines.........	57,425	7.12	204.43
Coarse.......	85,893	7.42	318.66
	143,318		523.09
829	79,916	8.92	356.43
834	81,210	8.91	361.79
	161,126		718.22
		After screening:	
Fines.........	58,396	9.83	287.02
Coarse.......	102,730	8.44	433.52
	161,126		720.54

Table IX.—*Sampling-Results, Copeland Sampling Co., Victor, Colo.*

Using Oscillating Time-Samplers.

Mill Mixes on Cripple Creek Gold-Ore:

Lot No.	Weight. Pounds.	Assay Gold. Ounces per Ton.	Gold. Ounces per Ton.
603	2,237	17.81	
	1,223	25.685	
	1,705	67.07	
	5,183	1.25	
	6,846	2.59	
	10,015	0.485	
	18,488	1.545	
Mathematical average,		5.322	Machine-sample of mix, 5.35
907	1,759	1.795	
	13,220	2.54	
	19,271	1.72	
Mathematical average,		2.04	Machine-sample of mix, 2.04
941	16,696	1.28	
	17,179	0.79	
	15,066	1.50	
	2,729	1.39	
Mathematical average,		1.187	Machine-sample of mix, 1.23

Lot No.	Weight. Pounds.	Assay Gold. Ounces per Ton.	Gold. Ounces per Ton.
976	7,645	2.80	
	11,117	1.97	
	2,828	6.69	
	2,899	4.925	
Mathematical average,		3.124	Machine-sample of mix, 3.12
669	18,005	1.83	
	22,534	1.48	
Mathematical average,		1.07	Machine-sample of mix, 1.62
791	8,254	4.93	
	10,130	2.38	
	8,346	2.08	
Mathematical average,		3.073	Machine-sample of mix, 3.12

TABLE X.—*Sampling-Results, Copeland Sampling Co., Victor, Colo.*

Using Oscillating Time-Samplers.

Cripple Creek Gold-Ore:

Lot No.	First Sample. Gold. Ounces per Ton.	Resample. Gold. Ounces per Ton.
260	14.065	13.96
270	1.01	0.99
606	0.56	0.54
639	0.59	0.60
692	1.28	1.30
707	1.30	1.25

The most convincing tests of correct valuation in ore-sampling are those in which numbers of small lots are bought and paid for individually, and stored for a considerable time, until a sufficient quantity of ore has been collected to form one large lot. When this period arrives the individual lots are not mixed, but run through the mill in succession, and it is usually found that the mechanical sample of the mixture agrees with the calculated average as determined by the values in the original purchases as closely as the best control-assays.

The small lots when originally received, sampled, and purchased were coarse and generally wet, but when run through the mill the second time they are both fine and dry, giving thereby the greatest possible dissimilarity in conditions of size of particles and moisture-content. The excellent checks ob-

tained on this class of work show conclusively that with "time-sampling" the results obtained are in no way affected by the physical conditions of the ore, and may be implicitly accepted as correct.

The art of sampling has now reached a stage where a standardization of methods is both desirable and possible, and it is to be hoped that the Mining Congress, or the proposed Bureau of Mines, will take the matter under consideration and appoint a thoroughly qualified commission which will give the subject the study and investigation its importance demands. Recommendations by an unbiased, competent board would do much to eliminate faulty methods, and bring about the adoption of standard systems of valuation which would prove of inestimable benefit to the mining and metallurgical industries from both a business and a scientific stand-point.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Assay and Valuation of Gold-Bullion.*

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(Spokane Meeting, September, 1909.)

THE Bureau of the Mint of the United States Treasury maintains 13 offices for the purchase of gold-bullion, and this paper describes an investigation to establish the reasonable differences in the assay-results at the various institutions which may be commercially allowable in the settlements between them. Beginning with the comparative assay of proof-gold at the Philadelphia mint and the Utrecht mint, which shows 0.00002 as the closest agreement now possible, eight tables of comparative results, taken from the regular work of the service, are given. These tables begin with very fine gold, produced in an electrolytic refinery, showing close agreement in the assay-results, and follow through decreasing gold fineness and increasing amounts and complexity of base metals to very impure and complex bars produced at cyanide-mills, some of which give widely-varying results. Next is given a series of results on samples, prepared and sent out to various laboratories in the service, to test the influence of different metals and various combinations upon the agreement of the assay-results; 11 samples were sent out and each one was assayed from 44 to 71 times, making a total of 623 assays. To these are added 107 assays of identical samples of coin-gold.

On a previous occasion,[1] I have endeavored to show the degree of accuracy that may be expected in the ordinary every-day analysis of various materials, and on another occasion[2] I

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[1] The Actual Accuracy of Chemical Analysis, *Trans.*, xxvi., 370 (1896); *Journal of the American Chemical Society*, vol. xviii., No. 9, p. 808 (Sept., 1896).

[2] The Accuracy of the Commercial Assay for Silver, *Journal of the American Chemical Society*, vol. xvi., No. 8, p. 505 (Aug., 1894); *Journal of the Society of Chemical Industry*, vol. xv., No. 6, p. 434 (June 30, 1896).

have called attention to the accuracy of the commercial assay for silver. The present paper deals chiefly with an effort to establish certain commercial standards of agreement or accuracy in the assaying of gold-bullion for purchase in the various laboratories of the U. S. Mint Service.

Besides the coinage-mints at Philadelphia, New Orleans, Denver, and San Francisco, and the large assay-office on Wall street in New York City, all of which purchase and refine bullion, the Mint Bureau of the U. S. Treasury maintains eight smaller assay-offices, scattered about through the mining-districts of the country, which purchase bullion and ship it to the mints to be refined. These smaller offices were established as an aid to the mining-industry of the country by giving the small miners the opportunity to sell their bullion easily and quickly for cash.

Owing to the particular and rigid methods of book-keeping of the Treasury Department, the mints are compelled to treat the bullion sent to them from the assay-offices in exactly the same manner as the bullion deposited directly by individuals, and the prices are carefully determined at which the assay-office bullion should be charged against the mint in the Treasury accounts. Naturally, discrepancies sometimes arise between the mints and the assay-offices. A very large proportion of these are small and are easily adjusted. In fact, most of them adjust themselves automatically, as they are on both sides of the account, and the gains and the losses over a period of time will counterbalance each other. On rare occasions, however, the differences require adjustment by umpire-assays in the laboratory of the Bureau of the Mint at Washington. For several years I have been gathering data upon the subject, and have had a series of assays made in order to establish standard limits of differences which might be considered as allowable on different classes of bullion.

The methods of assaying followed in the various institutions are substantially the same, and have grown up as the result of many years of experience, so that with careful work on pure bullions the results obtained at different institutions ought to agree very closely; but with impure bullions, that is, bullions containing other constituents besides gold and silver, the chances for variations in the results increase. The action of

different impurities varies widely. Only small amounts of some impurities induce excessive variations in the results, while comparatively large amounts of others have but little effect, and, on the other hand, a combination of several impurities in a bullion may be most disastrous to any agreement of the assay-results.

The bullion is handled in the same manner at all the institutions. It is weighed as received, and then melted. Generally, a simple melting with soda or borax, or both, is sufficient, but sometimes it is more or less refined in the pot. In the case of large melts, 1,000 oz. or over, or of very impure bullion, a small sample may be dipped or poured out from the well-stirred pot and granulated in water. The granulations are used for the assay-sample. In general, however, the metal is cast into bars, and these bars are chipped, top and bottom, to obtain the assay-samples. The bars are again weighed and the assays made, when the value of the deposit is calculated from this data. If, however, the various assays made on a deposit do not agree well enough to satisfy the assayer, the bar is remelted, with or without refining in the pot.

The determination of gold in ores by the fire-assay, when properly executed, is justly regarded as one of the most accurate of analytical methods. With ordinary care and an excellent bead-balance, 1 part of gold in more than 20,000,000 parts of ore can be readily and accurately determined. The determination of 1 part of gold in 5,000,000 parts of ore is very easily done. Until recently, however, it was rare for commercial ore-assaying to attain to the accuracy of 1 part in 5,000,000.

The ability to determine gold in ores with such great accuracy is due to the fact that very large amounts of ore, up to 0.25 kg., are taken for the assay, and on a high-grade button-balance the resulting bead can be weighed to 1/200 mg. In assaying bullion, however, such extreme accuracy is out of the question, for the simple reason that there is a limit to the amount of bullion that can be taken for the assay. To obtain the most accurate results the assay-sample must be weighed on the same high-grade balance on which is weighed the resulting cornet, and the sample also must be weighed with the same degree of care and accuracy as the cornet. Now, the load that

a very sensitive bead-balance will safely carry is generally limited to 1 g., and the amount of metal generally taken for a gold-bullion assay is 0.5 g., or one-half of the maximum load of the balance.

Another point in bullion-assaying which militates against extreme accuracy in the results lies in the fact that the cornet which is weighed is itself gold, and, in high-grade bullions, it is a very large part of the sample taken for the assay, so that even slight errors in the handling of the cornet, resulting in slight losses or gains in its weight, count heavily against the highest accuracy of the results.

About two years ago samples of proof-gold were exchanged between the Philadelphia mint and the Utrecht mint, and these samples were assayed in comparison with the utmost care at both institutions, with the result that the Utrecht proof was pronounced slightly the best by both mints. The difference in the results of the assays at the two places was only 0.00002. This is by far the most careful and exhaustive comparison of gold-bullion assays known to me, and undoubtedly represents the limit of accuracy at present attained by human skill in such work.

Table I. shows a series of results obtained by three assayers working in the same laboratory upon fine gold from an electrolytic refinery. Each assayer worked upon the same sample in each set of assays as averaged, the samples being cut from both the tops and the bottoms of the bars. While there is a possibility that there may be some difference in composition between the tops and bottoms of the bars, yet in such high-grade material as this any such difference must be slight, and eight tests upon the subject showed a maximum difference between the top and bottom of only 0.0001, which is considerably less than many of the differences between individual assays. On the whole, then, the figures may be taken as fairly representing the ordinary run of commercial work upon such high-grade bullion. It will be noted that in several cases the figures exceed 1,000, which is due, in part at least, to the high grade of the material. It may also be due in part to the presence in this electrolytic gold of unusual impurities in very small amounts. These data emphasize the necessity of averaging a large number of assays to get a satisfactory determination of the fineness in such very high-grade material.

TABLE I.—*Fine-Gold Assays.*

1.	2.	3.	Average.	1.	2.	3.	Average.
999.8	999.6	999.6		999.7	999.7	999.3	
999.5		1000.0	999.7	999.5		999.8	999.6
999.7	999.5	999.4		999.9	999.8	999.5	
999.8		1000.1	999.7	999.8		1000 0	999.8
999.7	999.5	999.6		999.6	999.7	999.6	
999.8		1000.3	999.8	999.9		1000.3	999.8
999.4	999.4	999.6		999.7	999.5	999.7	
999.7		1000 3	999.7	1000.0		1000.4	999.8
			999.7				999.8
	999.5	999.6			999.8	999 9	
1000.1		999.9	999.8	1000.0		1000.2	999.9
999.8	999.9	999.6		999.7	999.8	999 8	
999.5		1000.1	999.8	999.5		1000.3	999.8
999.8	999.8	999.7		999.9	999.9	999.6	
999.5		1000.2	999.8	999.7		1000.1	999.8
999.9	999.7	999.6		999.8	999.8	999.8	
1000.0		1000.3	999.9	999.9		1000.5	999.9
				999.8	999.9	999.7	
				1000.1		1000.4	999.9
				999.8	999.8	1000.	
				1000.1			999.9
			999.8				999.9

Table II. shows results obtained by various assayers in a single laboratory in assaying granulation-samples from a wide variety of bullion.

The figures given in Table III. are all taken from a single shipment and show the accuracy that can be obtained upon material of fairly uniform composition, being mostly gold and silver, with but little base metal present. This table shows, first, the results obtained at the assay-office where the bullion was originally purchased; and, second, the results obtained upon the same material when shipped to a mint. In some of these samples there is undoubtedly a difference between the tops and bottoms of the bars, but the figures show the agreement that may be expected between two institutions in arriving at the value of such deposits.

Table IV. gives the assays of 14 bars which were referred to the Bureau laboratory for adjustment, although the average differences between the mint and the assay-office were only slight.

The handling of bullion produced at mills using the cyanide-process of gold-extraction has given a great deal of trouble.

TABLE II.—*Miscellaneous Gold-Assays.*

Gold Fineness.				Silver Fineness.
0.4	0.4	0.3	0.1	997.5
0.5	......	0.4	0.4	862.5
2.8	2.8	2.8	2 9	955.0
6.1	6.2	6.3	6.3	970.0
11.0	10.9	11.1	10.9	888.0
12.0	12.1	12.3	12.3	805.0
17.0	16.5	17.1	17.0	967.5
19.4	19.4	19.7	19.6	835.0
29.4	29.4	29.2	29.2	709.0
36.1	36.0	35.9	36.0	689.0
43.2	43.2	43.1	......	47.0
47.0	46.4	45.4	46.3	304.0
52.4	51.8	52.0	51.4	79.0
62.5	62.7	62.1	62.5	766.0
68.0	68.2	68.3	68.0	362.0
79.0	79.4	79.0	79.1	731.0
108.5	109.3	108.7	108.9	495.0
148.3	148.4	148.3	148.4	372.0
179.0	179.1	179.0	....	695.0
194.1	194.3	194.7	195.0	771.5
208.3	208.3	208.6	....	416.0
308.4	308.8	308.5	308.6	149.0
439.9	440.0	439.8	440 0	190.0
510.1	510.0	509.6	509.6	236.0
515.0	515.1	514 9	515.2	171.0
537.7	537.8	536.8	537.1	227.0
571.6	570.4	571.4	571.0	185.0
605.3	606.9	606.7	606.8	129.0
642.6	643.0	643.8	642 7	257.0
711.2	710.2	710.7	711.7	3.0
716.0	716.1	716 0	715.9	222.0
758.9	759.0	759.0	759.1	216.0
870.6	870.5	870.2	871.4	27.0
978.0	978.4	978.0	....	17.0

TABLE III.—*Assays of a Single Shipment.*

	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.
Gold fineness.	843.4	843.0	860 9	860.3	862.5	862.6	863.6	863.4	864.8	864.6
	843.4	843.0	860.9	860.6	862.3	862.5	863.6	863.5	864.8	864.6
	843.4	843.2	860.8	860.6	862.5	862.5	863 6	863.4	865.0	864.5
	843.2	843.1	860.9	860.7	862.6	862.6	863.3	863.6	864.7	864.4
	843.2	843.0	860.9	860.7	862.5	862.5	863.6	863 6	864.8	864.9
	843.1	843.3		860.7	862.5	862.6	863.6	863.6	864.7	864.7
Silver fineness...	151.5		135		134		132		131	

	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.
Gold fineness.	870.5	870.8	873.5	873.2	874.9	874.9	878.2	878.1	880.1	879.9
	870.6	870.9	873.6	873.9	874.9	874 9	878.3	878.2	880.1	880.0
	870.6	870.9	873.6	873.6	875.1	874.7	878.3	878.3	880.2	880.2
	870.5	870.6	873.4	873.7	874.9	874 7	878.3	878.2	880.2	880.1
	870.6	870.8	873.5	873.5	874.7	874.7	877.8	878.2	880.0	880 2
	870 6	870.8	873.7	873.4	874.9	874.7	878.2	878.1	880.0	880.1
Silver fineness......	125		122		120.5		117		116.5	

Table IV.—*Comparison Between Assay-Office, Mint, and Bureau.*

	Gold Fineness.													
Assay-office.	736.6	807.5	850.2	853.1	866.8	868.3	876.4	878.2	879.1	884.0	886.0	892.8	897.4	899.5
	736.4	807.6	850.3	853.4	867.0	868.0	875.7	878.2	879.1	884.1	885.5	892.5	897.3	899.2
	736.5	......	......	......	866.5	868.4	......	......	......	884.0	......	......	......	......
	736.5	......	......	......	866.6	868.6	......	......	......	883.9	......	......	......	......
Mint.........	735.4	807.2	849.1	852.6	865.7	867.5	875.1	877.6	878.3	883.7	885.0	891.7	896.4	898.8
	735.4	806.7	849.4	852.7	865.2	866.9	875.4	877.6	878.6	883.0	885.4	891.9	896.2	897.8
	735.1	807.3	849.9	852.5	865.9	867.5	......	877.7	878.5	883.9	885.2	891.7	896.6	898.9
	735.9	807.1	850.2	852.5	866.3	867.5	......	877.8	878.8	883.6	885.3	892.0	896.8	898.0
Bureau......	735.9	807.4	850.1	852.7	866.8	867.9	875.9	878.2	879.2	883.7	885.7	892.6	897.2	898.7
	736.0	807.5	850.1	853.0	866.6	867.5	875.9	878.1	879.2	883.7	885.5	892.8	897.1	898.7
	736.1	807.5	850.2	853.1	866.7	867.8	875.8	878.3	879.4	883.8	885.9	892.1	897.1	898.6
	736.1	807.5	850.0	852.9	866.8	867.7	875.9	878.4	879.1	883.6	885.9	892.1	897.3	898.8
Highest.........	736.6	807.6	850.3	853.4	867.0	868.6	876.4	878.4	879.4	884.1	886.0	892.8	897.4	899.5
Lowest..........	735.1	806.7	849.1	852.5	865.2	866.9	875.1	877.6	878.3	883.0	885.0	891.7	896.2	897.8
Difference.......	1.5	0.9	1.2	0.9	1.8	1.7	1.3	0.8	1.1	1.1	1.0	1.1	1.2	1.7
Silver...........	179.0	181.0	133.0	142.0	127.0	112.0	120.0	117.0	115.0	113.0	109.0	88.0	99.0	95.0

Even when properly prepared such bars are likely to be troublesome, but when, as not infrequently happens, the slimes are not properly purified before being melted into bars they may give no end of trouble.

A very mild case of variation in cyanide-bars is shown in Table V. As received, these bars were chipped and the chips assayed. Since the figures thus obtained were considerably higher than the shipper's figures, the bars were then carefully bored and the borings assayed. Finally, the bars were remelted, with small losses in each case, and granulations taken. The granulations were then assayed.

TABLE V.—*Assay of Cyanide-Bars.*

	Gold Fineness.				
Chips.......... ...	394.1	381.6	380.7	381.6	440.8
	392.7	383.2	381.7	381.8	440.9
	392.0	381.3	381.5	381.5	440.3
	392.0	383.4	383.4	382.8	440.4
Borings...........	394.0	381.6	380.7	381.6	440.8
	392.1	381.3	381.5	381.5	440.3
Granulations.....	393.5	382.6	382.0	381.8	440.7
	393.4	382.3	382.7	382.5	440.4
	393.6	382.8	382.4	381.2	440.1
	393.8	382.2	383.2	381.4	440.1
Silver fineness........	370	370	370	370	357

Table VI. exhibits the results obtained by sampling three cyanide-bars, high in gold and very low in silver, in three different ways. The assays show a wide variation on the chip-samples. While the drill-sample assays are fairly concordant for this class of material, the dip-sample assays agree much better and are to be preferred.

An assay-office had received a cyanide-bar which showed 546, 545.5, 546.2, 546 fine in gold. This was considered satisfactory, and it was shipped to a mint, but the chip-samples there yielded most varying results, as follows: 544.6, 535.2, 543, 535, 542.4, 555.6. The bar was then remelted, and granulations showed 550.2 and 551.2. Another cyanide-bar received at the same assay-office from the same mill showed 592, 593.9, 592.9, 593.3 fine in gold, and was accepted. It was shipped to the same mint, where chips showed 603.6 and 590, while borings showed 588 and 588.6. The bar, which weighed 559.65

TABLE VI.—*Assay of Cyanide-Bars.*

Sampled in three ways.

	Gold Fineness.		
Chips	833.1	864.2	839.8
	828.1	863.3	841.9
	830.7	866.2	845.2
	842.1	869.6	839.1
Drills	834.5	864.2	845.5
	833.7	864.2	845.3
	832.7	865.8	845.0
	835.5	867.4	845.4
	835.4	867.0	844.4
	834.6	866.6	845.5
Dips	834.1	865.9	845.7
	834.8	866.6	845.1
	834.2	867.1	844.9
	835.3	865.1	845.5
	834.8	865.2	845.0
	834.1	866.9	843.4
	834.5		
	834.7		
	834.6		
	834.5		
Silver fineness	5.5	8.	8.
Weight	1169.06 oz.	1228.40 oz.	1171.16 oz.

oz. Troy, was remelted, with a loss of 1.78 oz., and granulations from the melt showed 601.8 and 601.8 fine in gold.

Having had a great deal of trouble with some bars from this mill, while others gave but little trouble, the assay-office gave one of the bad bars a very thorough treatment by melting and refining in the pot. As received, the bar weighed 643.30 oz. Troy, and was probably about 847 fine in gold. It was melted seven times, when it weighed 502.01 oz., showing a loss of 141.29 oz. in weight. The final bar was 933.4 fine in gold and 21 fine in silver. The gold-loss from this excessive course of meltings was only approximately 3.75 oz., most of which could undoubtedly be recovered from the slags.

The details of the meltings are shown in Table VII. It should be noted that the fourth melt shows practically no refining, and the weight was only slightly reduced, so that no practical change is shown in the assays.

TABLE VII.—*Cyanide-Bar, Melted Seven Times.*

Original weight, 643.3 oz. Troy.

	Gold Fineness.	
First melt, 557.22 oz.....	847.0	847.0
	847.2	846.6
	848.0	846.3
	847.8	844.6
	847.6	847.6
Second melt, 544.46 oz..	868.1	868.3
	867.1	868.8
	865.8	866.7
	866.1	866.6
	866.8	867.5
	869.2	869.1
	867.9	867.4
Third melt, 536.44 oz...	877.3	877.3
	877.3	879.4
	875.7	876.6
	873.8	875.8
	877.4	878.7
	876.7	877.3
	876.6	876.8
	875.4	877.5
	878.9	874.6
	879.2	874.1
	876.0	865.7
	875.4	875.2
	877.0	878.9
	877.8	879.4
	863.4	877.7
	875.7	876.9
Fourth melt, 535.55 oz..	878.3	878.7
	879.0	871.4
	877.5	861.6
	877.1	878.0
	875.8	879.5
	870.2	879.0
	876.1	877.5
	876.9	878.0
	871.7	875.7
	879.0	879.0
Fifth melt, 511.88 oz....	916.8	917.3
	916.9	917.6
	917.6	916.8
	917.1	916.9
Sixth melt, 504.82 oz...	928.6	928.6
	929.2	928.8
	929.4	928.6
	930.0	928.8
Seventh melt, 502.01 oz.	933.5	933.3
	933.2	933.4
	933.3	933.7
	933.7	933.4

From an extensive series of tests made at the San Francisco mint it was found that, as a rule, in the cyanide-bars from several California plants, the chip-samples taken from the outside of the bars would be about 2.5 fine less in gold than the borings

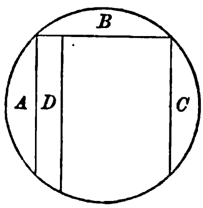

FIG. 1.—GOLD COIN SAMPLED FOR ASSAY.

when taken away from the edges of the bar, and that the borings gave satisfactory samples of the bars.

When made from the highest grade of metals our coin-gold, 900 gold and 100 copper, does not segregate. The gold used may contain a very small amount of silver, but should be as free as possible from all other impurities, and the copper should be

of the highest purity possible. Occasionally, in practice, however, there will be some segregation due to some impurities present in minute amounts. On one occasion an inside strip cut from a double eagle was assayed six times and yielded the following gradually decreasing figures: 900.2, 900.1, 899.9, 899.85, 899.5, 899.45. On another occasion a double eagle was cut as indicated in Fig. 1 and the following results were obtained:

	A.	B.	C.	D.
	Gold Fineness.	Gold Fineness.	Gold Fineness.	Gold Fineness.
Bureau	899.12 899.45 899.20 899.45	900.47 900.38	899.89 899.85	899.70 899.58
Mint	899.4 899.4 899.5 899.5	900.2 900.5 900.2		

One of our most annoying and yet very interesting and instructive cases was a lot of foreign-coin gold, the product of a mint which is very careful in the manufacture of its coins; 12 deposits of this material were received at the Philadelphia mint from the New York Assay-Office. It was supposed to be 916⅔ fine in gold, the balance being copper, and very uniform in composition, but the New York assays showed considerable variation. At Philadelphia one man assayed each deposit in duplicate, and he was checked by another man with a single assay, as shown in Table VIII.

TABLE VIII.—*Foreign-Coin Gold-Assays.*

	Gold Fineness.											
First assayer	917.6	916.4	916.4	917.6	917.5	917.1	915.6	917.3	917.6	914.7	917.0	917.0
	918.3	917.9	917.9	917.8	917.9	917.5	917.3	917.3	917.5	916.8	916.9	917.1
Second assayer	917.2	917.3	917.7	917.3	916.5	916.3	917.2	916.3	916.0	915.8	915.4	916.0

This bullion was diluted with copper to bring it down to the United States standard. While our standard is 900 fine, and the law allows a variation of one one-thousandth up or down, so that legally the coins may run from 899 to 901 fine in gold, yet the working-limits adopted at the mints are much narrower than the legal limits, and generally no gold ingots are passed by the assayer below 899.7 fine nor above 900.2 fine.

In making ingots from this metal an unusual number of melts had to be rejected and remelted for want of uniformity. It was expected that the coins made from this bullion would run low, but none of those regularly tested did. In fact, two from one delivery were most unusually high, viz.: 900.6 and 900.7. Thereupon 12 coins were selected from the same delivery and 46 assays were made upon them, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
3 assays showed . .	899.7	4 assays showed . .	900.0	5 assays showed . .	900.4
3 assays showed . .	899.8	7 assays showed . .	900.1	4 assays showed . .	900.5
5 assays showed . .	899.9	3 assays showed . .	900.2	1 assay showed . .	900.7
		11 assays showed . .	900.3	—	
				46	

The trouble with this metal undoubtedly arose from the presence of a small amount of some impurity causing a segregation of the gold, but enough work to decide what this was could not be given to the matter. In a similar case, with a different high-grade foreign-coin gold at the San Francisco mint, the trouble was traced to the presence of a minute amount of antimony.

In order to get a much wider range of comparison, and to test the influence of the different metals and of various combinations upon the gold-assay, a series of samples was prepared in the Bureau laboratory and sent out to various laboratories in the service for assay. In preparing the samples the metal was thoroughly mixed by stirring when molten and remelted as often as appeared necessary. They were finally cast into small bars, and when sufficiently ductile were rolled out thin. The strips were cut into small squares, and these were mixed up and the samples for each institution taken out of the mixed pile of pieces. In the case of the brittle bars, they were hammered out and rolled until they crumbled to pieces. The larger pieces were then cut up, and the whole mixed before the samples were taken out.

All through the preparation of the samples very great care was exercised, so that in each set every sample sent for assay should be identical, and thus eliminate from the assay-results all chances of differences being due to differences in the samples operated upon, and to confine the differences shown to the

actual assay-work. In one very base sample, which will be further noted, it was not possible to adhere to this rule because the metal was too hard.

In making such small melts it is practically impossible to adhere to any predetermined composition with any degree of closeness.

The first sample sent out was gold about 105 fine in silver and about 10 fine in copper; 71 assays of this sample were made in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
3 assays showed .	. 884.1	6 assays showed .	. 884.4	7 assays showed .	. 884.8
3 assays showed .	. 884.2	11 assays showed .	. 884.5	3 assays showed .	. 884.9
5 assays showed .	. 884.3	14 assays showed .	. 884.6	—	
		19 assays showed .	. 884.7	71	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
884.271	884.517	884.663
884.433	884.517	884.738
884.438	884.631	884.788

A sample approximately 500 fine in silver, 110 fine in copper, and 50 fine in lead was assayed 64 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
2 assays showed .	. 340.9	3 assays showed .	. 341.4	3 assays showed .	. 341.9
7 assays showed .	. 341.0	7 assays showed .	. 341.5	4 assays showed .	. 342.0
11 assays showed .	. 341.1	9 assays showed .	. 341.6	1 assay showed .	. 342.1
7 assays showed .	. 341.2	3 assays showed .	. 341.7	—	
2 assays showed .	. 341.3	5 assays showed .	. 341.8	64	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
341.016	341.163	341.600
341.038	341.467	341.863
341.150	341.520	341.913

Two samples were both about 25 fine in mixed base metals, while one was approximately 360 fine in silver, and the other was about 450 fine in silver. The first sample was assayed 61 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
1 assay showed . .	617.6	8 assays showed . .	618.1	4 assays showed . .	618.5
4 assays showed . .	617.7	6 assays showed . .	618.2	9 assays showed . .	618.6
3 assays showed . .	617.8	9 assays showed . .	618.3	2 assays showed . .	618.7
9 assays showed .	618.0	6 assays showed . .	618.4	—	
				61	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
617.725	618.233	618.388
618.025	618.283	618.467
618.138	618.320	618.480

The second sample was assayed 60 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
4 assays showed . .	528.6	4 assays showed . .	529.0	7 assays showed . .	529.4
7 assays showed . .	528.7	7 assays showed . .	529.1	2 assays showed . .	529.5
3 assays showed . .	528.8	12 assays showed . .	529.2	1 assay showed . .	529.6
3 assays showed . .	528.9	10 assays showed . .	529.3	—	
				60	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
528.671	529.175	529.267
528.800	529.238	529.283
528.963	529.250	529.300

Having on hand some ferruginous bullion, I attempted to prepare a sample for this work, but experienced considerable difficulty in getting a satisfactory metal, owing to the separation of magnetic globules on solidification. By melting several times with niter I finally obtained a sample that did not show visible segregation, and it must have been close to saturation with iron. It was about 763 fine in gold and 185 fine in silver, so that the entire base metals, including the iron, were only about 52 fine.

This sample was assayed 47 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
2 assays showed . .	762.9	6 assays showed . .	763.4	4 assays showed . .	763.8
5 assays showed . .	763.0	4 assays showed . .	763.5	5 assays showed . .	763.9
3 assays showed . .	763.2	2 assays showed . .	763.6	2 assays showed . .	764.0
5 assays showed . .	763.3	9 assays showed . .	763.7	—	
				47	

The averages obtained in the different laboratories were as follows:

Gold Fineness.	Gold Fineness.	Gold Fineness.
762.975	763.417	763.700
763.175	763.467	763.683
763.300	763.500	763.833

It having been supposed that much of the difficulty with cyanide gold bars was due to the zinc left in the slimes and going into the bars, a sample was prepared which was nearly 590 fine in gold, about 245 fine in silver, slightly over 130 fine in zinc, and containing a little copper and very little lead.

This sample was assayed 50 times in eight laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
1 assay showed . .	588.9	7 assays showed . .	589.3	2 assays showed . .	589.7
3 assays showed . .	589.0	7 assays showed . .	589.4	5 assays showed . .	589.8
4 assays showed . .	589.1	9 assays showed . .	589.5	3 assays showed . .	589.9
3 assays showed . .	589.2	6 assays showed . .	589.6	—	
				50	

The averages obtained in the different laboratories were as follows:

Gold Fineness.	Gold Fineness.	Gold Fineness.
589.040	589.417	589.567
589.400	589.475	589.800
589.400	589.483	

A simple inspection of these results shows very clearly that zinc alone does not materially militate against agreement in the assay-work itself, and if it is the cause of the trouble with cyanide-bars it must be owing to its causing segregation, and thus preventing the proper sampling of the bars by chipping or boring. Other elements may also be active in producing segregation in such bars, either by themselves or through combinations with the zinc or other metals present. A low-grade and very base bar along this line was prepared to run about 100 fine in zinc, 200 fine in copper, and 50 fine in lead. It was about 268 fine in gold and 370 fine in silver. This bar was very hard, and it was impossible to prepare identical samples for the various laboratories. It was simply cut into pieces and a piece sent to each institution.

This sample was assayed 44 times in eight laboratories, and while the difference between the highest and the lowest result is only 1.7 fine, yet the results are scattered all along through the range, and there is only a slight concentration of the results about one point. This is, of course, due in part to the fact that the samples assayed were not identical.

The results obtained were:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
1 assay showed . .	268.0	3 assays showed . .	268.5	3 assays showed . .	269.1
3 assays showed . .	268.1	3 assays showed . .	268.6	1 assay showed . .	269.3
6 assays showed . .	268.2	3 assays showed . .	268.8	3 assays showed . .	269.6
6 assays showed . .	268.3	1 assay showed . .	268.9	3 assays showed . .	269.7
4 assays showed . .	268.4	4 assays showed . .	269.0	—	
				44	

It has long been known in a practical way that the presence of arsenic in a gold-bullion prevents any agreement in the assays. Fortunately, however, the presence of arsenic very plainly reveals itself in the melting of the bullion, and when found the melter proceeds to refine the bullion in the pot, and ultimately removes it very completely before the bullion can be accepted.

Three test-samples containing arsenic were prepared, and they yielded most astonishing and interesting results. The first sample was approximately 785 fine in gold, 107.5 fine in silver, 100 fine in copper, and 7.5 fine in arsenic. This is only a small proportion of arsenic, and yet it completely prevented any agreement whatever in the assay-results. This sample was assayed 65 times in ten laboratories. The lowest result obtained was 779.7 fine in gold, and the highest 792.4, with an extreme difference of 12.7 in the fineness. Moreover, there is the utmost divergence in the results as well as no agreement whatever; 30 results were obtained only a single time each, 11 only twice each, 3 only three times each, and only a single result was obtained four times. In only three instances did one laboratory obtain the same result twice.

A sample approximately 675 fine in gold, 200 fine in silver, 100 fine in zinc, 24 fine in lead and copper, and only 1 fine in arsenic, yielded a trifling better set of results, but still very widely divergent. This sample was assayed 62 times in ten laboratories. The lowest result obtained was 671.4 fine in gold,

and the highest 681.4, showing an extreme difference of 10 in the fineness; 31 results were obtained a single time only, 10 only twice each, 2 only three times each, and only a single result was obtained five times. In three instances one laboratory obtained the same result twice, and in one case a laboratory obtained the same result three times.

It would appear, however, that the influence of arsenic upon the assaying of high-grade bullion containing only trifling amounts of base metals is far less injurious. While the results on a sample running approximately 865 fine in gold, 130 fine in silver, 1 fine in arsenic, and only 4 fine in other base metals cannot be considered satisfactory, yet they are very much better than those yielded by the other two arsenical bullions. This sample was assayed 53 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
1 assay showed . .	864.1	4 assays showed . .	865.2	3 assays showed . .	865.9
1 assay showed . .	864.3	8 assays showed . .	865.3	5 assays showed . .	866.0
2 assays showed . .	864.4	2 assays showed . .	865.4	2 assays showed . .	866.1
2 assays showed . .	864.7	4 assays showed . .	865.5	3 assays showed . .	866.2
2 assays showed . .	864.8	1 assay showed . .	865.6	1 assay showed . .	866.5
2 assays showed . .	865.0	3 assays showed . .	865.7	—	
3 assays showed . .	865.1	4 assays showed . .	865.8	53	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
864.933	865.233	865.500
865.183	865.286	865.517
865.200	865.300	865.717

As in so many other directions, antimony behaves similarly to arsenic in assaying gold-bullion, but its influence is not so pronounced. A sample of bullion approximately 723 fine in gold, 245 fine in silver, 1 fine in antimony, and 31 fine in mixed base metals, copper, lead, zinc, was assayed 46 times in nine laboratories. The lowest assay obtained was 721.3, and the highest 725.1, showing a range of 3.8 in the fineness. However, 24 of the results, or just over a half, ranged from 722.8 to 723.9 fine, and outside this range only two results were obtained more than a single time.

Finally, some of our gold coin was melted up and assayed

107 times on identical samples in five laboratories, with the following results:

	Gold Fineness.		Gold Fineness.
6 assays showed . . .	899.6	32 assays showed . . .	900.0
10 assays showed . . .	899.7	5 assays showed . . .	900.1
26 assays showed . . .	899.8	—	
28 assays showed . . .	899.9	107	

The actual average of this sample is 899.879 fine in gold.

With these results as a basis, the investigation of the subject is being continued with the hope of ascertaining the causes of the variations shown and improving the agreement in the results attained. It is, for instance, well known that gold cornets are not pure gold. They always carry some silver, and I have never failed to find copper in them when tested for with great care. On several occasions I have found lead present on testing the silver nitrate solution from parting a large number of cornets at one time in a platinum basket. The amounts of these base metals present in the cornets are, of course, quite small, and their influence is corrected by the proof-assay, in the same way that it corrects for the silver left in the cornets. I am, however, carrying on a series of quantitative determinations of base metals present in gold cornets, the results of which I hope to publish at some future date.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Laws of Fissures.

BY BLAMEY STEVENS, SEATTLE, WASH.

(New Haven Meeting, February, 1909.)

THE object of this paper is to present a theory of the formation of fissures which seems to be supported by all available data. The investigation is, in the main, an exact one, and irregularities of the rock-structure are generalized. From the conclusions drawn a general classification of fissures is made, and this should be of practical use in the comparison of mineral deposits. The theory also throws some light on the equilibrium of the earth's crust and the depth of action of surface-waters. Incidentally, additions are made to the theory of earthquake-slips.

Former Theory.

It has been previously pointed out that the "jointing" of homogeneous rocks occurs in two sets of planes, at right angles to one another, and making angles of 45° with the directions of greatest and least principal stress.[1]

The theory by which this fact is explained does not, however, account for fissures and faults, because normal faults are usually more vertically, and reverse faults more horizontally, inclined than 45°.

The "jointing law" is based on the following axiom: "In a homogeneous mass under pressure, rupture must take place on the lines of maximum tangential stress."[2] This law applies, with fair approximation, to the tests on jointing which can be made in the laboratory, and as nearly as can be expected to the natural jointing-planes in rock-masses, where the stresses are not very great; but it entirely fails to account for fissure-fractures.

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[1] G. F. Becker in *Bulletin of the Geological Society of America*, vol. iv., p. 48 (1892). See also, Torsional Theory of Joints, *Trans.*, xxiv., 130 (1894).

[2] *Ibid.*

Author's Revised Theory.

It is generally assumed that these joint-planes are fissures and faults in embryo, but we shall show in this paper that an entirely different law governs the formation of faults and fissures. This law may be conveniently known as the fissure law, and may be stated as follows:

In a homogeneous mass under pressure, slipping tends to take place only along those planes on which the ratio of tangential stress to direct stress is equal to the coefficient of friction of the material sliding on itself.

The law is proved by observations on actual rock-masses and by laboratory-experiment.

Exact Analysis.

The condition of stress in any one place in a rock-mass or other solid substance may be completely represented by three principal stresses (p_x, p_y, p_z) at right angles with one another. (See Fig. 1.)

Let us assume $p_x > p_y$ and $p_y > p_z$; and in Fig. 1 let the plane of the paper be at right angles to the medium principal stress, p_y. Now let us consider the action of forces and stresses on a plane, GH, which is also supposed to be perpendicular to the plane of the paper. The stresses on any plane must be clearly distinguished from the forces on it, the stress being the amount and direction of force per unit-area of plane. If p_x and p_z represent principal stresses (as indicated by the equally-spaced arrows), the total force parallel to p_x on the plane GH is $GH \times y \times p_x \sin\theta$, where y is the breadth of the plane GH, measured parallel to the y axis. This may be resolved into $GH \times y \times p_x \sin^2\theta$ normal to the plane GH, and $GH \times y \times p_x \sin\theta\cos\theta$ tangential to it. Similarly, the total force parallel to p_z on the plane GH is $GH \times y \times p_z \cos\theta$, and this may be resolved into $GH \times y \times p_z \cos^2\theta$ normal and $GH \times y \times p_z \cos\theta \sin\theta$ tangential. Hence, the total normal force is $GH \times y\,(p_x \sin^2\theta + p_z \cos^2\theta)$, and the total tangential force is $GH \times y\,(p_x \sin\theta\cos\theta - p_z \cos\theta\sin\theta)$. As all these forces are distributed over an area $GH \times y$, the total normal stress is

$$\left.\begin{array}{l} p_x \sin^2\theta + p_z \cos^2\theta \\ \text{or } \dfrac{p_x + p_z}{2} - \dfrac{p_x - p_z}{2}\cos 2\theta, \end{array}\right\} \qquad (1)$$

and the total tangential stress is

$$\left.\begin{array}{l} (p_x - p_z)\sin\theta\cos\theta \\ \text{or } \dfrac{p_x - p_z}{2}\sin 2\theta. \end{array}\right\} \qquad (2)$$

It may be easily seen that expression (2) forms a maximum when $\cos 2\theta = 0$, *i.e.*, when $\theta = \pm 45°$. These are the rectangular joint-planes formed when the tangential stress (p_t), necessary to rupture, is reached, *i.e.*, when

$$\frac{p_x - p_z}{2} \sin 2\theta = p^t. \tag{3}$$

Or, putting $\theta = \pm 45°$, we get $\pm \dfrac{p_x - p_z}{2} = p^t$. At the same time the normal stress is

$$\frac{p_x + p_z}{2}.$$

Now if a coefficient of friction be assumed at unity, slipping can only begin when the tangential and normal stresses are equal, *i.e.*, when $\pm (p_x - p_z) = p_x + p_z$. This is only possible when $p_x = 0$ or ∞, or $p = 0$ or ∞, neither of which is a possible condition; that is to say, with a coefficient of unity joint-planes cannot possibly form faults.

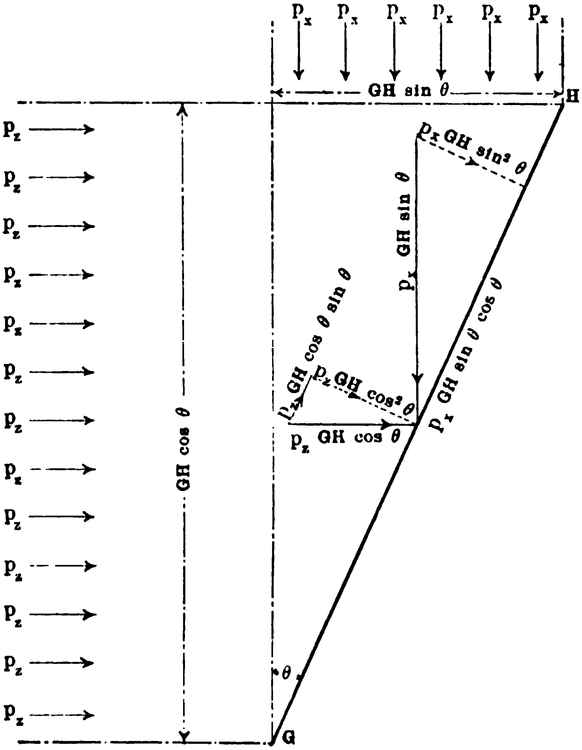


FIG. 1.—DIAGRAM ILLUSTRATING THREE PRINCIPAL STRESSES IN A ROCK-MASS.

Let us now proceed to find the inclination of the fracture which is most favorable to the formation of fissures or faults. In order that slipping may occur between the surfaces of any fissure, the rules of frictional stability must be obeyed; that is to say, the coefficient of friction multiplied by the normal component of stress must be equal to the tangential stress.

In symbols, $a\,(p_x \sin^2 \theta + p_z \cos^2 \theta) = (p_x - p_z) \sin \theta \cos \theta$ (4)
where a is the coefficient of friction.

It is evident that there is a ratio between p_x and p_z which will vary as θ and a are varied; this ratio is

$$\frac{p_x}{p_z} = \frac{\sin\,\theta\,\cos\,\theta + a\,\cos^2\theta}{\sin\,\theta\,\cos\,\theta - a\,\sin^2\,\theta} = \frac{1 + a\,\cot\,\theta}{1 - a\,\tan\,\theta} \qquad (5)$$

One of the principal stresses is supposed to be vertical; its mean value must, on the average, be dependent on and equal to the weight of overlying material; and under a certain range of conditions more fully dealt with elsewhere in this paper, one of the principal stresses, which is horizontal, will tend to be as great or as small as is consistent with occasional slipping along fissures when the limiting ratio of principal stresses is reached. This limiting ratio depends on a and θ, and has a maximum with respect to θ which will be found to be given by the equation $\cot\,2\,\theta = a$. If a be fixed at $\pm\,0.9$, which seems to be the mean value most in conformity with observed data, then

$$\theta = \pm\,24°.$$

Result of Analysis.

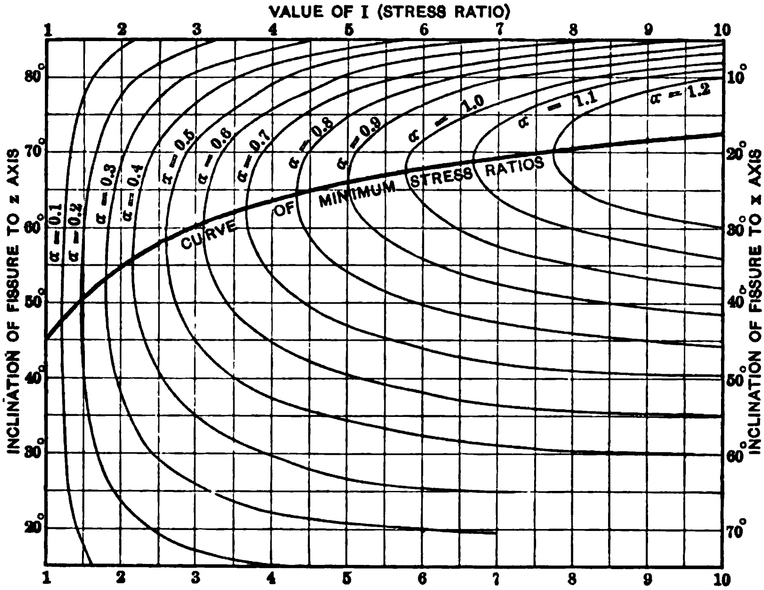
What we have demonstrated in the foregoing mathematical argument is that, if there be fractures at all inclinations in a rock-mass, then, with a coefficient of friction of $a = 0.9$, slipping will occur only along those inclined at 24° to the axis of greatest principal stress. It will be seen from the diagram, Fig. 1, that if the axis of greatest principal stress is vertical, the displacement along the fissure will be that of a normal fault, and the dip of the normal fault which is most favorable to slipping will be 90 — 24 = 66°. Similarly, when the axis of greatest principal stress is horizontal, the displacement along the fissure is that of a reverse fault, and the dip most favorable to slipping is 24°.

TABLE I.—*Ratios of Greatest to Least Principal Stresses Necessary to Produce Slipping in Fissures.*

Inclination to Least Principal Stress (90—θ).	Coefficient of Friction (Statical). 0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8	0.9	1.0	1.1	1.2	Inclination to Greatest Principal Stress (θ).
90°	∞	∞	∞	∞	∞	∞	∞	∞	∞	∞	∞	∞	0°
85°	2.16	3.32	4.54	5 78	7.01	8.30	9.56	11.0	12.2	13.6	15.0	16.4	5°
80°	1.60	2.21	2.85	3 51	4.20	4 91	5.67	6.44	7.09	8.20	9.00	9.89	10°
75°	1.41	1.85	2.31	2.80	3.42	3 87	4.43	5.07	5.76	6.47	7.24	8.10	15°
70°	1.32	1.67	2.04	2.45	2.91	3.38	3.92	4.52	5.15	5.89	6.72	7.65	20°
65°	1.27	1.58	1.91	2.27	2.71	3.18	3.71	4.34	5.03	5.88	6.89	8.10	25°
60°	1.24	1 52	1.84	2.21	2 63	3.12	3.70	4.45	5.33	6.48	7.96	10.0	30°
55°	1.23	1 50	1.81	2.18	2.64	3.19	3.92	4.87	6.18	8.09	11.2	23.2	35°
50°	1.22	1.48	1.82	2.22	2.75	3.46	4.46	5.96	8.45	13.6	30.0	∞	40°
45°	1.22	1.50	1.86	2.33	3.00	4.00	5.67	9.00	19.0	∞	∞		45°
40°	1.24	1.54	1.95	2 55	3.51	5.27	35.4	∞	∞				50°
35°	1.25	1.60	2.12	3.00	4.74	9.90	∞						55°
30°	1.28	1.71	2.45	4.00	9 61	∞							60°
25°	1.33	1.92	3.20	8.37	∞								65°
20°	1.43	2.39	6.37	∞									70°
15°	1.64	4.15	∞										75°

This table is constructed from formula (5). The following example illustrates its use:

It is wished to find the horizontal stress which has caused slipping in a normal fault, dipping at 75°, the coefficient of friction being estimated at 0.7 and the mean depth at 10,000 ft. The greatest principal stress will be vertical and the least principal stress horizontal, so that the dip will be the same as the inclination to the least principal stress. Opposite 75° in the first column and under the coefficient of friction (0.7) we find the figure 4.43. The least horizontal stress is therefore equivalent to a depth of rock of $\frac{10,000}{4.43} = 2,257$ feet.



This diagram corresponds with Tables I. and II. The curve of minimum stress ratios gives the most favorable angle of slipping for any given coefficient of friction.

FIG. 2.—DIAGRAM SHOWING ISO-FRICTION CURVES AND CURVE OF MINIMUM STRESS RATIOS.

Table I. and the corresponding diagram, Fig. 2, show the dips of fissures consistent with any coefficient of friction (α) and ratios of stress $\left(\frac{p_x}{p_z} \text{ or I.}\right)$

A larger ratio of the principal stresses will cause slipping, but for equilibrium the ratio I must be smaller than the one indicated.

It will be seen from the diagram that in general there are two angles at which slipping will just occur; but for a minimum stress-ratio these two become coincident, and the corresponding angle is then known as "the most favorable angle of slip" for the particular coefficient of friction.

If the stress-ratio is reduced below the minimum no slipping can occur at any angle with the given coefficient of friction. Conversely, as the stress-ratio of a rock-mass fractured at all inclinations is increased from a low equilibrium value, slipping takes place at "the most favorable angle of slip."

A fissure well lubricated by talc would have a smaller coefficient of friction than 0.9; in this case the most favorable dip of the fissure might be less than 66° or more than 24°.

Complete Problem.—Let us now try to find how the stresses in a solid rock-mass could break a new fissure along a plane which would be favorable to combined breaking and slipping. We will presume that the tangential stress on the plane along which fracture is occurring is just sufficient to overcome the sum of friction and fracture. Referring to expressions (1) and (2), we therefore construct the following equation:

$$\frac{p_x - p_z}{2}\sin 2\theta = a\left(\frac{p_x + p_z}{2} - \frac{p_x - p_z}{2}\cos 2\theta\right) + p_t \qquad (6)$$

The ratio $\frac{p_x}{p_y}$ then becomes a maximum when $\cot 2\theta = a$. This is exactly the same result as when simple sliding without fracture was considered (see Table II.).

Let us now examine the theory in the light of experimental tests made in the laboratory by the application of simple crushing-tests. In these cases $p_z = 0$, but by our assumption (from equation 6)

$$p_z = \frac{p_x(\sin 2\theta + a\cos 2\theta - a) - 2\,p_t}{\sin 2\theta + a\cos 2\theta + a}. \qquad (7)$$

or putting $\theta = 24°$ and $a = 0.9$

$$p_z = \frac{0.2227\,p_x - p_t}{1.1227}. \qquad (8)$$

If p_x is measured by the depth, or head of rock causing the pressure, then p_t, in equivalent units, will be 1,600 ft. for granite.

In Fig. 3, the strong lines are constructed from equation (8) with $p_t = 1{,}600$ ft., whence it will be seen that when $p_z = 0$ then $p_x = 7{,}200$ ft., which corresponds very closely to compression rupture as determined by testing-machine.

Similarly, the dotted lines in the diagram are constructed from equation (3), viz.: $\frac{p_x - p_z}{2} = p_t$, whence, putting $p_z = 0$ we get $p_x = 3{,}200$ ft. This figure also corresponds very closely with the "cracking" noted by observers in the same laboratory crushing-test.

The above investigation shows how the fracture of fissures may take place, and that this theory of fissures is supported by the results of laboratory compression-tests of rock-specimens, if $\alpha = 0.9$. If θ is the inclination of the fracture to the greatest principal stress, then $\cot 2\theta = \alpha$, gives the value of θ which is the most favorable to slipping, and from experimental tests this seems to correspond very closely with the value of θ which is most favorable to fracture.

Table II. and the thick line in Fig. 2 show the value of the stress-ratio, I, for all values of θ or α, for either fracture or the most favorable slipping-conditions.

TABLE II.—*Inclinations and Stress-Ratio Constants for Combined Fracture and Slipping of Fissures.*

Coefficient of Friction. α.	Inclination to Axis of x. θ.	Inclination to Axis of z. $90-\theta$.	Constant $I = \frac{p_x}{p_z}$ when $p_t = 0$.	Constant $B = \frac{p_x}{p_t}$ when $p_z = 0$.
0.0	45° 00′	45° 00′	1.00	2.0
0.1	42° 09′	47° 51′	1.22	2.2
0.2	39° 21′	50° 39′	1.48	2.4
0.3	36° 39′	53° 21′	1.81	2.7
0.4	34° 06′	55° 54′	2.18	3.0
0.5	31° 43′	58° 17′	2.62	3.2
0.6	29° 31′	60° 29′	3.12	3.5
0.7	27° 30′	62° 30′	3.68	3.8
0.8	25° 40′	64° 20′	4.34	4.1
0.9	24° 00′	66° 00′	5.00	4.5
1.0	22° 30′	67° 30′	5.8	4.8
1.1	21° 08′	68° 52′	6.7	5.2
1.2	19° 54′	70° 06′	7.6	5.5

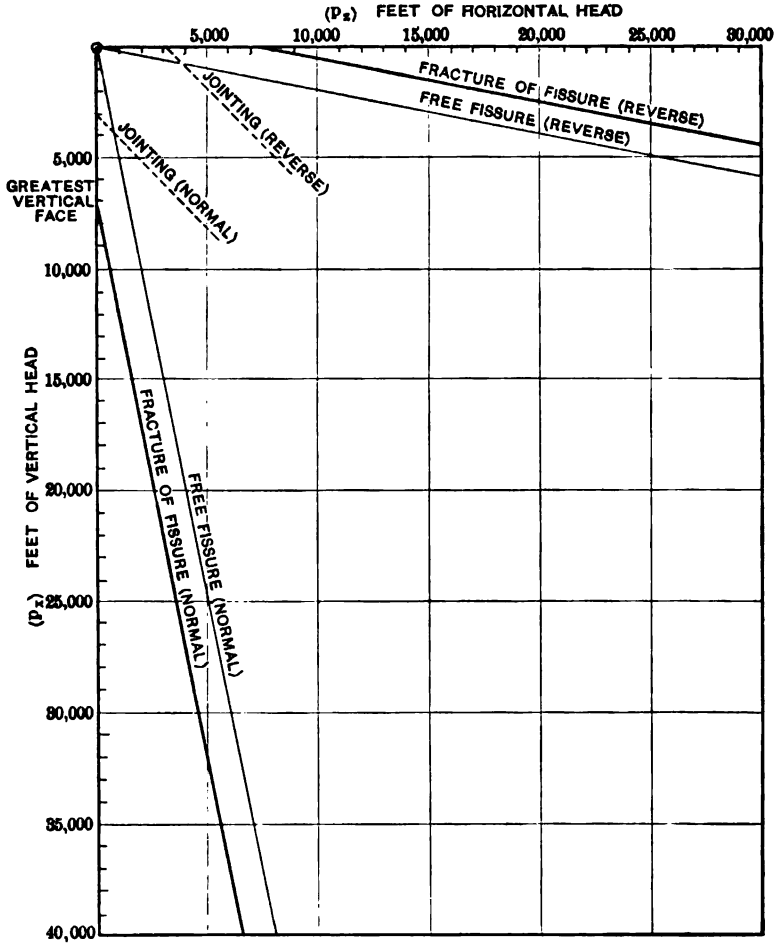
NOTE.—The formula for fracture making use of above constants is

$$p_x = Ip_z + Bp_t.$$

Fig. 3 shows the greatest horizontal stress compatible with fracture when the coefficient of friction is 0.9 and the greatest principal stress is vertical. The theoretical dip of the fracture is 66°, the same dip as that previously determined as the most favorable to slipping between the walls of a normal fault. The dotted line shows the greatest horizontal stress compatible with cracking or the formation of joint-planes dipping at 45°. The diagram illustrates also the comparative unimportance of fracture, as compared with friction, by the proximity to one another of the lines of fracture and free fissures.

Limitations of Jointing.

The phenomenon of jointing belongs essentially to brittle matter; in tough substances there is either no jointing at all or it manifests itself at angles of less than 45° with the greatest principal stress. The inference is, that friction interferes with



Constants: $p_t = 1{,}600$ ft. head; $\alpha = 0.9$.

FIG. 3.—VERTICAL AND HORIZONTAL STRESS DIAGRAM FOR AN AVERAGE GRANITE.

fracture before fracture actually occurs, as we have assumed in equation (6), which we should expect to be nearer the truth for tough rocks than for brittle ones. It would also seem that rock-stresses may become too great for jointing to be formed.

Elimination of Cohesion.

The laws of sand-equilibrium as deduced by Rankine[3] are a special case of the rock laws as discussed in this paper, and are obtained from them by equating to zero the tangential stress necessary to fracture the rock. It will be noticed that this condition is also approached when the actual rock-stresses are very large; so that at considerable depths this tangential resistance may be disregarded, and we may illustrate the rock as acting like so much sand, held together only by gravity and friction. Considering the smallness of the simple tangential stress necessary to fracture rock in comparison with the evident great thickness of the earth's crust, we must therefore conclude that the earth as a whole owes no noticeable amount of its stability to cohesive forces. In this connection we find that Lord Kelvin, in his calculation that the earth must be more rigid than steel to resist the distortion due to the tidal forces of the sun and moon, uses the term "rigid" only in the sense that sand is rigid under gravitational forces and friction, and he does not presume any cohesion to exist.

Varieties of Fissures.

Let us now consider what differences there may be in the character of a fracture near the surface and at great depth. Some amount of energy must be necessary to fracture. Near the surface, this energy will be considerably in excess of that required to start a slip on a fissure previously existing. The fractured surfaces will therefore tend towards a minimum of area; consequently, they will be clean cut and not multiple. On the other hand, at great depths, where the cohesional forces are of no relative importance, the rock will be always ready to form new fissure-courses, and each of these may be a complex and multiple fracture. Intermediate between these limiting forms of fissures we have every gradation. Various terms have been employed for these features, but the following three are perhaps sufficiently distinctive.

Varieties of Fissures.	True or clean-cut fissures, formed at small depths. Fissures with false walls, formed at medium depths. Shear-zones, formed at great depths.

The plastic metamorphism evidenced in the structure of

[3] *Applied Mechanics.*

many rocks may also be elucidated in part by the comparison with sand, but other mechanical, physical, and chemical forces undoubtedly have their influence on the change effected.

It is evident that the depth at which the above varieties of fissures are found in a homogeneous formation is proportional to the tangential stress necessary to fracture, expressed in linear measurement head of rock; thus, in sandstone a certain variety of fissure would be formed at from a half to a quarter of the depth at which the same variety would be formed in granite.

Classes of Fissures.

Let us now find how the ratio of vertical to horizontal thrust in the rock-mass adjusts itself. There are certain external conditions to consider. Chief among these is gravity, the action of which justifies us in assuming that over a moderate area of country, we may depend upon the mean vertical component of stress, at any level, being a direct stress, and represented by the depth or head of rock (H) above the level considered.

The horizontal stresses will be either greater or less than the vertical stress or depth (H). If less, the least of them must not be more than $\frac{H}{I}$ (where I is the ratio shown in Table II.) for any adjustment by slipping to take place. The solvent action of the surface-waters circulating through the cracks and crannies of the rock-mass tends continually to shrink the whole mass. This relieves the horizontal stress until it cannot balance the vertical stress H, because it has fallen slightly below $\frac{H}{I}$. Slipping then occurs, and the horizontal stress is thereby raised by the wedge-like action of the portion of rock-mass descending in the manner known as normal faulting. The horizontal stress remains in this latter state only until solvent actions or possibly some other causes relieve it.

If the greatest horizontal stress is greater than the vertical stress or depth H it cannot be less than IH for any adjustment to occur. When slipping comes it tends to relieve the horizontal strain by the wedge-like motion of the two portions of the rock-mass approaching each other horizontally, and one of them being pushed up vertically in the manner known as reverse faulting.

Besides normal and reverse faults, we may have side-thrust faults, formed by the greatest and least principal stresses being both horizontal. The sliding is then horizontal and the fissure vertical. The San Francisco earthquake fissure-slip of 1906 was of this class, the greatest principal stress being about true north and south and the fissure running about N. 25° W.

We may also place in a separate class those faults in which none of the principal stresses are vertical.

We thus arrive at the following four classes of fissures:

Class.	Greatest Principal Stress.	Least Principal Stress.
Normal.	Vertical.	Horizontal.
Reverse.	Horizontal.	Vertical.
Side-thrust.	Horizontal.	Horizontal.
Skew.	Having no vertical axis of stress.	

Regions of Fissuring.

From the above classes of fissures we may divide the earth's crust into four regions, viz.:

A. The region of normal fissures.

B. The region of no fissures.

C. The region of side-thrust fissures.

D. The region of overthrust fissures.

Region *A* consists of rocks near the surface, where the solvent action of percolating waters or some more deep-seated disturbance makes normal fissures possible. This region is important as being the one we are most intimately associated with, and in which the richer concentrations of valuable mineral occur. In this region the least horizontal stress is not greater than $\frac{H}{I}$. If it is a little less, owing to there being no nearby fissure of proper angle, there may be a side-thrust fissure movement which really belongs to region *B*.

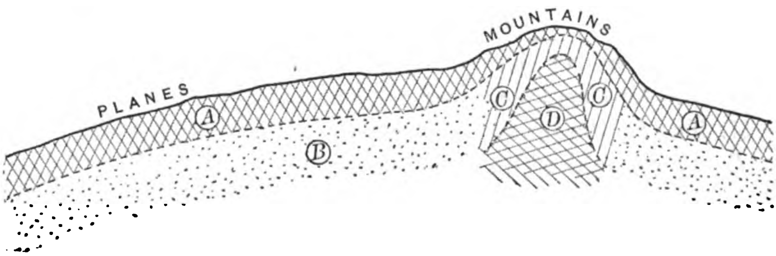
In regions *B* and *C* the least horizontal stress is more than $\frac{H}{I}$ and the greatest horizontal stress is less than IH. In region *B* the ratio of greatest to least horizontal stress is less than I, and in region *C* this ratio is greater than I. It is quite probable that there is no fissuring at all in regions which underlie the great plains to an enormous depth. Reasoning from this standpoint, it would seem that beneath the plains

region A would correspond to a region of surface-water circulation. This deduction may throw some light on the much-disputed depth of action of surface-waters. Around the mountains we should expect some rocks to be subject to side-thrust movements, and the region B to give place to region C, which is of considerable importance in mining-districts.

In region D the greatest horizontal stress is not less than IH. This region becomes prominent near the earth's lines of weakness. The overthrust faults here force up large masses of rock into mountain ranges.

Skew fissures do not fall into any regional division because the requirement of a region is that the vertical stress shall be direct.

The diagram, Fig. 4, illustrates the probable distribution of the three regions and one extra sub-region in both plane and mountain country.



A. **Region of normal fissures.** *B*. **Region of no fissures.** *C*. **Region of side-thrust fissures.** *D*. **Region of reverse fissures.**

FIG. 4.—SHOWING DISTRIBUTION OF FISSURE-REGIONS.

Earthquakes.

Slipping is not a slow, creeping process, but a series of sudden jars. This is explained as follows:

During the process of slipping we have, in order to be exact, to consider the changes in the value of the coefficient of friction, α. Before motion actually starts we must use the statical coefficient, α_0, but during the process of slipping we must use the dynamical coefficient, α_1, which is less than α_0. The coefficient α_0 determines the value of the horizontal stress when motion starts and α_1 determines its value when it stops, except that some allowance should be made for the extra stress which tends to arrest motion. The effect of the sudden slipping over a considerable distance is an earthquake, which is due solely to the difference in value of α_0 and α_1. In all

problems of fissure-equilibrium which we have considered by exact rules the statical coefficient of friction, α_0, is the only one we should take into consideration. The great horizontal stresses are no doubt formed by the arched condition of the crust and its relief from some of the support of the central core. This results from the diminution of the core by loss of material (*i. e.*, by volcanic and other emanations), and in a less degree, perhaps, by loss of heat.

Other things being equal, and α being 0.9, the violence of fissure earthquakes in the region of reverse fissures will be twenty-five times as great as those in the region of normal fissures. In the region of side-thrust fissuring, the violence will be from five to twenty-five times as great as in the region of normal fissures.

Influence of Third Axis.

When the third axis, y, with its corresponding principal stress, p_y, is taken into account we shall find that the maximum ratios of tangential to normal stress occur when $l = 0$ or $m = 0$ or $n = 0$, where l, m, n are the direction cosines of the normal to the plane under consideration. By comparison of these maximum values we find that if $p_x > p_y$ and $p_y > p_z$ the greatest maximums occur when $m = 0$. These are the cases we have examined in assuming the paper to be perpendicular to the y axis and, therefore, all the maximums we have so far considered are grand maximums; that is, they represent the most favorable conditions for jointing, fracturing, or slipping.

There are two special cases, however, which need our consideration, viz.:

$$\text{when } p_x > p_y = p_z \text{ and when } p_x = p_y > p_z$$

In either case the elypsoid of stress becomes circular about what we may term the "odd" axis. The resultant fracture will make a definite angle with this odd axis, but there is no longer any reason why it should be definitely inclined to either of the other two axes, which we may call a "pair" of axes. This is the kind of fracture known as conchoidal. When there are small irregularities in the texture of the rock the plane of fissure will be somewhat modified, and when p_y is not exactly intermediate between p_x and p_z the fissure, although having the average strike and dip of a plane, will have conchoidal modifications locally displayed. The resultant fracture will make a more definite angle with the odd axis than with either of the other two, but the general average plane of the fissure will still include the intermediate or y axis.

Sub-Classes of Fissures.

We have previously considered only two principal axes of stress, viz., the axis of greatest stress and the axis of least stress. There is, however, of necessity, a third axis which may have any stress value between the greatest and least stresses. The general strike and dip of a fracture are not affected by the

value of this intermediate principal stress, but when it is nearly equal to either of the other principal stresses, there becomes some question as to which of these two nearly equal stresses shall determine the direction, and the fracture seems to wabble to and from the mean course determined by the greatest and least stresses. If we call the "odd axis" the one which is least inclined to be equal in value to either of the others, which we may call a more or less perfect "pair," we may say that the resultant fracture will make a more definite angle with the odd axis than with either of the other two. This angle may, in fact, be more definite than though there were no axis appreciably odd. The surface of fracture which will be formed when a perfectly homogeneous medium is acted on by two exactly equal principal stresses and one odd one is that made up of elements of a cone on the odd axis. If, however, the intermediate axis is not exactly equal to one of the other axes, we may expect the fissure to tend towards a series of parallel corrugations whose length corresponds with the direction of slip. With a corrugated fissure it will not, in general, be possible to ascertain whether the greatest or the least axis of stress is the odd one; hence we cannot make as exact a classification as theory calls for. Unlike sinuous corrugations, conical surfaces of any considerable curvature can only form an extensive fissure by being linked together and forming a linked vein. With a linked vein there should, in general, be little difficulty in determining the odd axis.

The above cases may be classified as follows:

Plane fissures, . . .	no odd axis or pairing.
Corrugated fissures,	imperfect pairing.
Linked fissures, .	perfect pairing.

Secondary Displacement.

Secondary displacements may make an angle with the primary displacement and therefore with the corrugations, and so open up channels in which solutions can freely flow; even though the displacement is direct (that is, in the direction of corrugations formed at fracture), the wall-surfaces are more or less irregular along the corrugations, and channels are opened up.

Secondary-displacement "furrows" are often plowed up,

which gives us evidence of the direction of secondary displacements. This furrowing should not be mistaken for corrugation, which is usually formed on a larger scale than furrows. Fissures may thus be often divided according to their secondary displacement

into:
- Direct, . . . secondary displacement approximately along the corrugations.
- Indirect, . . oblique secondary displacement at an angle with corrugations.
- Square, . . . secondary displacement approximately at right angles to the corrugations.

Local Factors.

Displacement in an irregular fissure requires that the stress be confined to only a portion of the surface of the fissure, where it is correspondingly more intense than the mean stress; sub-jointing and sub-fissuring adjacent to the fissure may thus take place. At the same time, the grinding together of the walls takes off some of the irregularities, and supplies talc and other material to lubricate the movement of the fissure. All this gives rise to a more or less brecciated and pulverized filling-material between the two walls of the fissure, which, being now of conceivable breadth, we term a vein. The filling-material may be to a greater or less extent dissolved, replaced, and cemented together by mineral-bearing solutions, but this is a separate branch of the subject, the important mechanical consideration leading up to it being that a fissure arrived at the vein stage and filled with brecciated material offers the best conditions for the flow of solutions through deep-seated rocks. Under great stresses large open spaces cannot exist.

The pores of a close-grained, highly-compressed rock, such as is ordinarily met with in metal-mining regions, are very poor conveyors of mineral solutions, hence the interstitial spaces in brecciated material offer the only means by which such solutions may move about until conditions favorable to mineral deposition are reached.

In all cases, local weaknesses or irregularities are important factors in fixing the positions and extent of corrugations, furrows, and links.

Stratification or primary lamination not only makes the tan-

gential fracturing-stress different along different planes, but it also makes the elasticity of the rock different in different directions.

Sudden changes in the formation through which a vein is

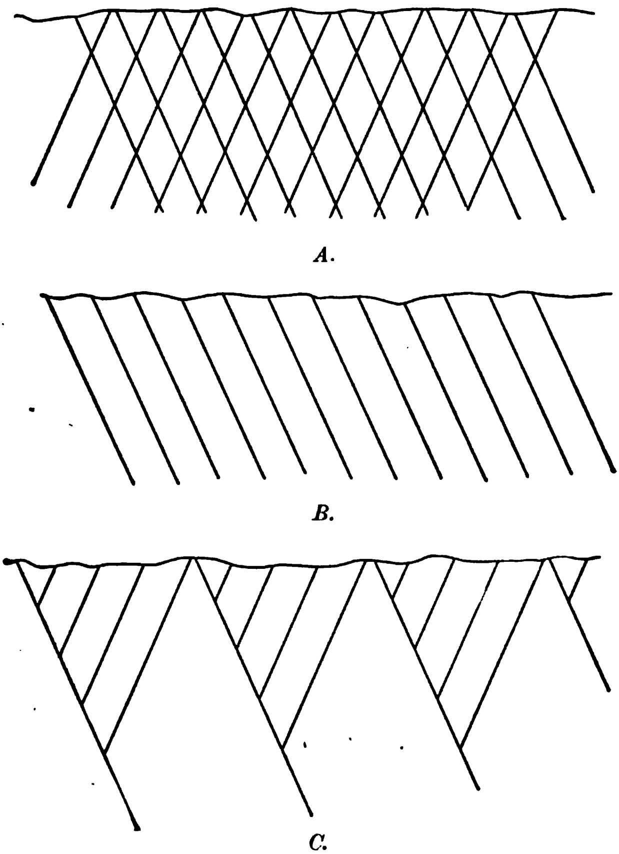


Fig. 5.—Systems of Fissures.

traveling are often noticed to be accompanied by equally sudden changes in the strike or dip of the vein.

Fissures in slates are sometimes a series of steps going first along and then with the formation.

Fissure-Systems.

The most complete system of fissures to be imagined as being formed by one condition of stress is a double system of parallel planes, inclined at $\cot^{-1} a$ to one another, as shown in Fig. 5 *A*.

As, however, a displacement on any one of these planes would preclude subsequent displacements on the planes which intersect it, we are in general limited to a single set of parallel planes, as shown in Fig. 5 *B*, or such a system of meeting, but not intersecting, fissures as is shown in Fig. 5 *C*. Intersecting systems of fissures must be attributed to the existence of different conditions of stress at different times.

A classification of fissures is given in Table III. A vein of any class may fall into any sub-class, and similarly with secondary displacements and varieties, so that there are 108 separate divisions into which a fissure may be placed without specific measurements being made or cognizance taken of the filling-matter.

TABLE III.—*Classification of Fissures.*

Class.			Sub-Class.		Secondary Displacement.		Variety.	
Designation.	Greatest Principal Stress.	Least Principal Stress.	Designation.	Pairing of Axes.	Designation.	Angle with Primary Displacement.	Designation.	Depth of Formation.
Normal.	Vertical.	Horizontal.	Plane.	None.	Direct.	None or Small.	True.	Small.
Skew.	Inclined or horizontal.	Horizontal or inclined.	Corrugated.	Imperfect.	Oblique.	Some odd angle. The value of this angle may be designated in figures.	False-walled	Medium.
Overthrust.	Horizontal.	Vertical.	Linked.	Perfect.	Square.	Right angle or nearly so.	Shear-zone.	Great.
Side-thrust.	Horizontal.	Horizontal.						

The further perusal of the "Laws of Fissures" may be advanced by taking the strains into consideration as well as the stresses. This may be the subject of another communication if there seems to be a demand for it.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Dust-Explosions in Coal-Mines.

BY FRANKLIN BACHE,* FORT SMITH, ARK.

(Spokane Meeting, September, 1909.)

THERE seems to be in the public mind, and even in the minds of some coal-operators not experienced in mines subject to dust-explosions, a feeling that there has been something mysterious at the bottom of a number of recent American colliery-explosions. It has been declared in cases of accidents in mines regarded as particularly well equipped, in which every preventive precaution was said to have been taken, that the explosions were incomprehensible, and resulted from causes beyond the present knowledge of the practical coal-operator. But it is safe to say that no explosion has taken place which could not be explained by reasons well understood by most operators.

The U. S. Geological Survey Test Laboratory, recently inaugurated at Pittsburg for the reported purpose of investigating scientifically the matter of explosions in mines, will discover no new explanation of explosions. It can, however, co-ordinate the previous investigations of similar boards in England and on the Continent; and it may do a vast amount of good by promulgating widely, and in a form comprehensible by the most unlettered miner, certain facts and obvious deductions that cannot but have an effect in reducing the number and extent of the mine-disasters which, in the last few years, have been so terrible in frequency and magnitude.

The only possible sources of explosions of any magnitude in coal-mines are: (1) explosives stored in quantity in the mine; (2) gases generated or liberated in the mine; and (3) coal-dust. Combinations of any of the three may, of course, take part in the result. The danger of any great loss of life from stored explosives alone is very remote. The simplest ordinary precaution forbids the presence, in one place underground, of any considerable quantity of explosives, and it would require an

* President Bache-Denman Coal Co.

exceedingly large quantity to cause such an explosion as would kill any considerable number of men, distributed through the mine. (The effect that firing a quantity of explosives would have in stirring up and igniting the dust is another matter.)

Inflammable gas, and its explosive mixture with air, the old enemies of the miner, will doubtless always constitute a lurking danger; but in the years since Davy invented his lamp we have learned pretty completely the ways of fire-damp. With modern fans and the widely-diffused and almost exact knowledge of ventilation, fire-damp is not allowed to accumulate; and even in mines making gas so fast that explosive mixtures might be formed throughout the workings in a short time through any failure of ventilation, such failure is invariably guarded against by duplicate ventilating-plants. While the danger from gas is always present, we thoroughly understand it, and are taking all precautions that such a thorough knowledge suggests. The chance of gas-explosions affecting whole mines is very small. The danger to the individual miner, or to small numbers working in some section of a mine, is inherent in the work; and, do what we can with safety-lamps and well-directed air-currents, individuals here and there will be burnt. The effect of explosions of small quantities of gas in agitating and firing coal-dust, and the effect of gas in less than explosive proportions in the presence of a blown-out shot in a dusty mine, are again matters to be considered under the head of dust.

Coal-dust, therefore, alone remains as an indispensable cause of great explosions. For many years this source of danger has been recognized, particularly in the deep mines of Great Britain; but it was not until the investigation of the Pocahontas explosion in 1884 that the subject aroused more than theoretical interest in this country. Very little has been done here in the way of scientific and laboratory inquiry into this matter, probably because the investigations on the other side had been thorough, and there did not seem to be much room left to better them. Recently the U. S. Geological Survey has undertaken, through a competent commission, to make an investigation which will concern itself greatly with coal-dust as an explosive agent. But practically we know already about coal-dust all that we need to know. What we want to find out

is the best way of getting rid of the dust. We are quite aware that dust in a coal-mine is dangerous, and that the only way to make a mine safe in that respect is to eliminate the dust. We know that, next to eliminating the dust, the best thing is to undercut and wedge down the coal, using no explosive whatever, and that, if this be impracticable, the next best thing is to undercut the coal and use very little in the way of explosives, and that little as flameless and smokeless as it can be.

For reducing the danger from blown-out shots disturbing and firing the dust, we know only one means—namely, to undercut the coal, so that small charges of powder will suffice, and not "let the powder do the work." But it is difficult to see how we can get the coal undercut by the miner so long as he receives for a ton of coal, nearly all slack, the result of "shooting off the solid" a mighty blast of powder, the same pay as he receives for a ton of coal, nearly all lump, the result of undercutting the coal and using a small charge of powder.

So long as the miner gets as much per ton of "mine-run" coal with 80 per cent. of slack, which he has shot down from the coal-seam with the aid only of an excessive and dangerous amount of explosive, as he gets for a ton of "mine-run" coal containing 80 per cent. of lump, the result of first undercutting the coal and then letting the powder do only some of the work, he (being human) will "shoot off the solid," and when he finds that the enormous charge of powder is dangerous to his life he will endeavor to make coal-mining safe (to him) by insisting that the company shall procure men who do not mind the risk, to fire the shots. If the company does not see the matter in that light, the State legislature is induced to pass laws forbidding miners to fire their own shots, and compelling operators to employ other men to perform that dangerous duty.

If, in addition to procuring the passage of laws requiring the employment of other men to fire shots which the miners are afraid to fire themselves, the latter would go a step further and get additional laws passed prohibiting the firing of any shots unless the coal had been undercut, as a protection for the lives of the shot-firers and the safety of the property, we would have few dust-explosions, and the shot-firer could no longer truthfully say, when told that his wage of $3 for two hours' work was pretty high, "The company don't pay me no wages. I just

bet the company $3 a night that I won't get out of the mine. When the company loses, it pays me $3; when it wins, it buries me."

I have heard no rumor that any such combination of laws has been advocated by the miners before any State legislature. If such laws should ever be passed, it is needless to say that there will have to be an adjustment between the miners, the company, and the consumer as to how the cost of undercutting the coal shall be divided. One thing is certain: the undercut so much to be desired will not be made for nothing.

But pending the arrival of that semi-millennium we are not going to get rid of the miner's delectable sport of "shooting off the solid," with its frequent windy shots; and we shall continue to let the powder "do the work," which sometimes it surely does, as many a fatherless family can testify. Meanwhile, the miner has reduced his occupation from one requiring a considerable amount of skill, to one in which 90 per cent. of his labor consists of absolutely nothing but shoveling coal from the floor of the mine into a pit-car, a thing which could be done with a few hours' instruction by the most ignorant immigrant the day after he lands for one-third of what is paid by the "scale" of wages to the average miner of to-day.

Since we do not seem likely to get rid of the blown-out shot, we must devote ourselves to getting rid of the dust. The natural first step, as I have shown, would be to make less dust in mining the coal by undercutting it and thus getting mine-run coal containing, say, only 30 per cent. of slack, as compared to "shooting off the solid" and getting 60 per cent. of slack; but being unable to arrange that matter with our miners, let us consider what to do with the dust we actually have, whether it be necessarily or unnecessarily made.

The remedy almost universally suggested and employed is to water or otherwise dampen the dust. Laying pipe in the entries and watering the roadways thoroughly at frequent intervals is completely satisfactory if thoroughly carried out.

But the difficulty lies right there. The watering must be done at frequent intervals, and the pipe must be kept up to the face of the entries and kept in repair and renewed from time to time. This work becomes a matter of daily routine. The mine-foreman cannot give it his personal supervision at all

times. To lay the dust thoroughly is a long and rather tedious job, as any one will testify who has seen how water will apparently run under the dust, run around it, float it—in fact, do almost anything but mix with it. Moreover, the most impalpable and most dangerous dust will be found lodged on the ribs and roof. In many mines, to wet the roof means to bring it down, with attendant damage and expense.

Scattering in the entries some hygroscopic substance, such as calcium chloride or common salt, has proved fairly efficient.

Moistening the air with steam or a spray of water before it enters the mine is not effective.

In the summer, when the outer air is warmer than the mine, the cooling of the air as it enters the mine quickly causes it to deposit, on the roof and elsewhere, near the intake, the surplus moisture which its lowered temperature no longer permits it to carry. Hence, a humid air-current cannot be taken through the whole mine. To add moisture to the air of the down-cast would be useless.

In the winter, no matter how fully saturated the air may be as it enters the mine, as soon as its temperature is raised to that of the workings it is no longer saturated, and, instead of depositing, it absorbs moisture. In winter, therefore, unless the temperature of the entering air be raised to that of the mine at the same time that it is saturated with moisture, it would be useless to moisten it.

It might pay to heat the air to the mine-temperature at the same time it is moistened (both could be done by steam-jets, and exhaust steam now wasted could be used), or the air could be moistened in the mine after it had traveled far enough to acquire the mine-temperature.

But both watering the entries and moistening the air are subject, in many localities, to the very serious objection that the moistened roof will slack and fall, and become another source of danger and expense.

The best way to stop dust-explosions is to remove the dust, certainly from all parts of haulage-ways that are near enough to working-places to be affected by blown-out shots. It is not, in my opinion, necessary to remove the dust from rooms. The dust made in mining and loading the coal is comparatively coarse and safe. The dangerous material is the impalpable dust

made by the continual grinding by passing men, mules, and cars on the haul-ways.

If we get the dust out of the haul-ways whenever it is in juxtaposition to possible blown-out shots, we shall do away with 90 per cent. of the danger from dust-explosions. If we allow no shots to be fired except when all the employees but the shot-firer are out of the mine, we shall greatly reduce the loss of life in case of an explosion. And if, in addition, we can prohibit the firing of any shots in coal not undercut, we shall have almost no dust-explosions at all.

Sweeping the roof, sides, and floor of an entry for a few hundred feet and loading the dust out every week or two is not an appalling task. And the providing of dust-proof pit-cars would not be impossible. We could even do away with the end-door, and dump the car even on self-dumping cages by making some changes in our dumping-arrangements. And some time, who knows, we may get the miner to see that it is his interest, as well as ours, to put a premium on mining more lump- and less slack-coal.

The suggestion has been made that we would have fewer dust-explosions if we reduced our ventilating-currents. This is doubtless true. The less ventilation the less drying-out of the dust and the less dust stirred up. But it has taken a century to get all hands connected with coal-mines thoroughly imbued with the desirability of ample air, so that we now have no great disasters resulting from gas. Let us not compromise and decide that, in order to save some lives from dust-explosions, we will sacrifice more to gas-explosions. The logical thing is to reduce in every possible way the making of dust, and remove from the mine that which is unavoidable.

I trust that the Geological Survey Commission, in emphasizing the danger of firing heavy charges of explosives, will make its reasons so clear, express them so concisely and simply, and publish them in so many languages as to reach the understanding of every miner, even the most ignorant. At the same time, I trust it will avoid technical expressions, so that its conclusions and reasons may be readily understood by the non-mining public and by legislators. Under the pressure of public opinion, and through their own enlightenment, the miners themselves may consent to legislation limiting the charge of explo-

sives and compelling the undercutting of coal, and may agree to receive less pay for the comparatively valueless slack they produce and more pay for the lump, on a basis just alike to them and to the operator.

Should the Geological Survey Commission accomplish this, or make any material step towards its accomplishment, it will have justified its existence a thousandfold. Incidentally, a great diminution in the millions of tons of slack now yearly made would do much to conserve the national resources, and by reducing mining-fatalities to conserve the greatest natural resource of all—the lives of workers.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Residual Brown Iron-Ores of Cuba.

BY C. M. WELD, NEW YORK, N. Y.

(New Haven Meeting, February, 1909.)

ATTENTION has been turned recently to the exploration and development of certain large blanket-deposits of brown iron-ore in Cuba. The most conspicuous of these to-day, and the one upon which the most light has been shed, is the Mayari deposit, situated about 15 miles south of Nipe bay. Here the Spanish-American Co. has sole control over 18,500 acres of ore-bearing lands, reported by its engineers to contain 500,000,000 tons of ore; and the necessary plant and equipment, with docks and railways, is now under construction for the early marketing of this ore. A similar deposit, and undoubtedly the next to be exploited, is the ore-field at Moa bay, where 13,000 to 15,000 acres of ore-lands, immediately adjacent to the shores of an excellent harbor, have been generously covered by numerous mining-claims, practically all controlled by four large interests. This deposit is now estimated to contain approximately 350,000,000 tons, on the basis of dried ore ready for shipment, a figure which may be increased when the western limits of the ore-deposit have been more accurately defined. Other deposits of the same type, but smaller and less accessible, are those at Cubitas, situated from 12 to 15 miles north of Camagüey city, and at Taco bay and Navas, points lying a few miles west of Baracoa. The area of the Cubitas deposit is said to be 6,000 acres, and the yield of ore is estimated at 150,000,000 tons. The Baracoa deposits are less well known, but preliminary estimates have placed their joint ore-reserves at 40,000,000 tons.

Accepting the above tonnages as reasonably correct, we conclude that the deposits enumerated give promise of adding about 1,000,000,000 tons of iron-ore to the world's supply; they have, therefore, to be considered in any attempt to forecast the future of the iron and steel industries.

The map, Fig. 1, shows the approximate location of all the deposits mentioned. An illustrated description of the Mayari deposit, and the proposed plant and equipment for its exploitation, has already been published;[1] also, additional information, together with a brief reference to the Moa deposit.[2] These two papers are largely commercial in their attitude. A. C. Spencer, in his paper entitled, Three Deposits of Iron Ore in Cuba,[3] gives valuable and interesting information along more

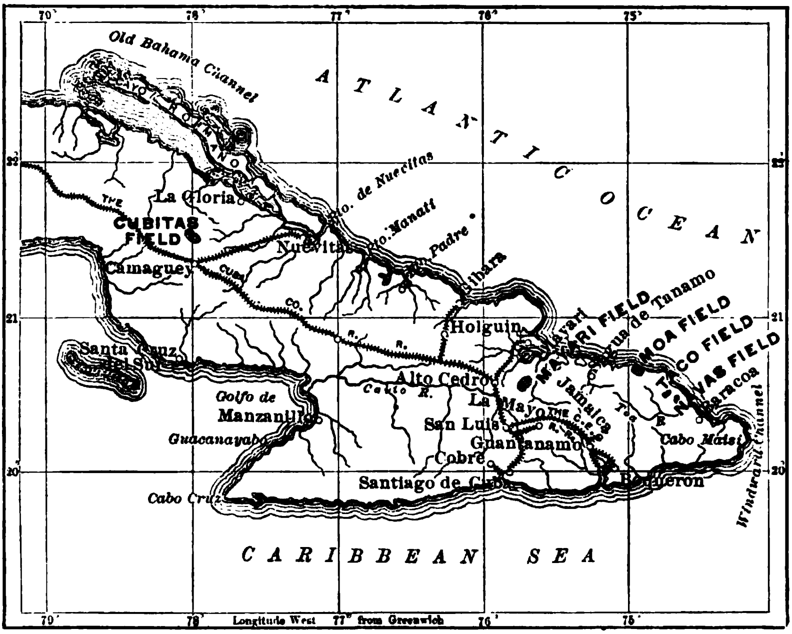


FIG. 1.—MAP OF PROVINCE OF SANTIAGO DE CUBA, SHOWING DEPOSITS OF BROWN IRON-ORE.

purely scientific and technical lines regarding the deposits at Mayari, Moa, and Cubitas.

While the subject of the present paper is therefore not altogether new, it has appeared to me that certain features concerning the character and probable genesis of the iron-ore deposits have not yet been brought out, and it is with this in view that the paper has been prepared.

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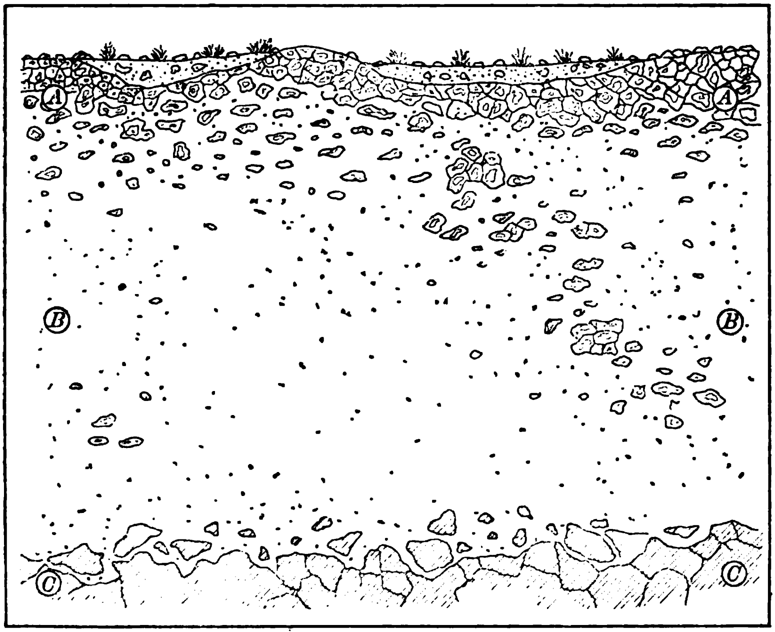
[1] *Iron Age*, vol. lxxx., No. 7, pp. 421 to 426 (Aug. 15, 1907).

[2] *Ibid.*, vol. lxxxi., No. 15, pp. 1149 to 1157 (Apr. 9, 1908).

[3] *Bulletin No.* 340, *U. S. Geological Survey*, pp. 318 to 329 (1908).

The deposits under discussion possess essential characteristics in common. They occur as residual mantles of enormous surficial extent, with a thickness occasionally as great as from 50 to 60 ft., but more commonly varying from 10 to 20 ft. The underlying rock is serpentine. The ore, which extends from the grass-roots to bed-rock, is for the greater part a homogeneous, tenacious, clay-like material, red to yellow to brown



A. Recemented capping of brown ore, from 1 or 2 up to 10 ft. thick. Frequently absent.

B. Clay-ore, red, yellow, or brown in color, from 7 or 8 up to 50 or 60 ft. thick. Contains disseminated nodules and pellets of brown ore, at times agglomerated in the form of beds or layers.

C. Serpentine bed-rock.

FIG. 2.—IDEALIZED VERTICAL SECTION SHOWING NATURE OF OCCURRENCE OF RESIDUAL IRON-ORE IN CUBA.

in color. The transition between ore and the comparatively unaltered serpentine bed-rock is as a rule fairly abrupt. Within the clay-ore are found disseminated nodules and pellets of brown ore ranging apparently through all the hydrated forms from limonite to turgite; hematite also is present, and at times magnetite. These concretionary forms increase in abundance towards the top of the ore-bed, where they frequently appear

as recemented masses of spongy brown ore, occasionally of large dimensions, forming beds or layers within the clays. At many places a capping or crust, as it were, of recemented brown ore of considerable extent is found at the immediate surface.

Fig. 2 represents an idealized section of the ore-beds from grass-roots to bed-rock. This sketch has been constructed from numerous observations in the field, and presents graphically the conditions above described.

Table I. gives representative analyses from four different fields. That for Mayari is taken from the article cited above,[4] while the others are of samples taken in connection with private examinations and not before published.

TABLE I.—*Analyses of Brown Iron-Ores of Cuba.*

	Mayari.	Moa.	Taco.	Navas.
Fe	46.03	46.75	46.23	42.48
SiO_2	5.50	1.71	2.06	3.01
Al_2O_3	10.33	11.60	2.16	6.12
Cr	1.73	1.81	2.07	2.39
TiO_2	*a*	0.14	*a*	*a*
P	0.015	0.031	0.021	0.032
H_2O	13.62	13.15	*a*	*a*

[a] Element not reported.

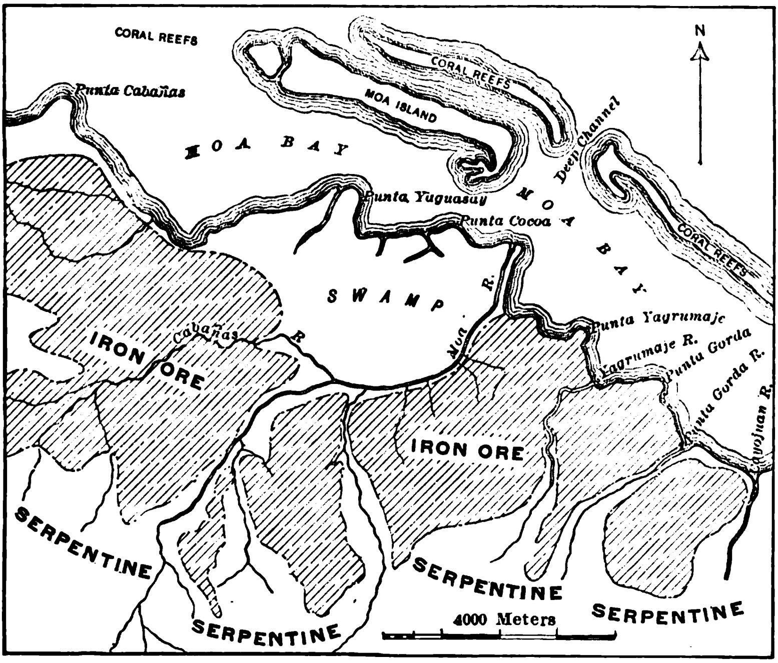
These analyses represent the entire mass of ore; that is, the clay-ore as well as the concretions and nodules. The figures are for the ores dried at 212° F. The ore in the ground frequently contains in addition up to as high as from 30 to 35 per cent. of hygroscopic moisture.

The general similarity of these ores is at once apparent, the conspicuous features being low silica, high alumina, and the presence of chromium in very appreciable amounts. These features are more especially true of the ores from Mayari and Moa, and it should be noted that the analyses given for those fields are the averages of a much greater number of samples than for the Taco and Navas fields; hence they should be regarded as more truly representative. In addition to the elements given above, nickel has been found in amounts of from 0.5 up to, in some cases, 2.0 per cent. Manganese is also present. The balance of the ore comprises small quantities of mag-

[4] *Iron Age*, vol. lxxx., No. 7, p. 424 (Aug. 15, 1907).

nesia and lime, with probably traces of the alkalies, and a very little sulphur.

Having established a common relationship between the ore-beds of these various localities, the discussion in its bearing upon questions of genesis will henceforth be more particularly confined to the ores at Moa, since my opportunities for personal observation have been confined to that field. The Moa



The cross-hatched area indicates approximately that occupied by iron-ore. The western extremity of the deposit is not shown, but has been taken into account in estimating the tonnage of ore offered by the field.

FIG. 3.—SKETCH-MAP OF THE MOA IRON-ORE FIELD, SANTIAGO PROVINCE, CUBA.

occurrence, moreover, as will appear later, approaches more nearly to a model type; at the same time, it will be seen that conclusions based on the Moa occurrence are equally applicable in their general aspects to the other occurrences. Before leaving the question of common relationship it may be well to review briefly the points from which this has been deduced, which are: (1) the blanket- or mantle-form of all the ore-beds; (2) the common bed-rock of serpentine; (3) the common

appearance and nature—namely, ocherous clay-like materials carrying disseminated nodules and masses of brown ores; and (4) the common analysis, showing low silica and high alumina, with notable quantities of chromium and much combined water.

The map, Fig. 3, shows the general features of the Moa deposit. The ore-bed occupies practically the entire area adjacent to the shore, and extends thence inland for a distance of from 3 to 5 miles, finally fingering out along the crests of the divides. Wherever the ore comes to an end laterally, and where it has been cut through by streams, serpentine is found as the country-rock. A number of samples of this serpentine were taken, of which an average analysis is given in Table II., together with a reconstructed complete analysis of the ore.

TABLE II.—*Analyses of Country-Rock and Iron-Ore, Moa Bay, Cuba.*

	Serpentine. Per Cent.	Iron-Ore. Per Cent.
Fe_2O_3	......	66.90
FeO	8.55	......
SiO_2	37.29	1.71
Al_2O_3	1.33	11.60
TiO_2	0.28[a] (TiO₂, Cr₂O₃, NiO together)	0.14
Cr_2O_3		2.65
NiO		0.60
P_2O_5	0.07	0.07
MnO	trace.	0.80
CaO	0.29	2.38[a] (CaO, MgO, K₂O, Na₂O together)
MgO	36.53	
K_2O	trace.	
Na_2O	0.39	
H_2O	15.27	12.15
	100.00	100.00

[a] Calculated by difference.

The ores have already been described as "residual mantles," the inference being that they have been derived from the underlying rocks by processes of sub-aërial decay. Any other theory of origin involves transportation.

The theory that they are derived directly from the serpentine is supported by the analyses in Table II. The ore is, in fact, a laterite, a product due to the peculiar form of decomposition known as laterization, which is common to humid tropical climates. The essential characteristic of laterization is the breaking-up of the silicates, with the ultimate almost com-

plete removal of the silica, wherein it differs radically from the kaolinization-processes of the temperate zones. Laterites have been reported from many tropical localities, and are especially common in India.

The ordinary procedure in rock-decay involves the removal of lime, magnesia, and the alkalies, while the aluminous silicates and the ferric oxides for the greater part remain behind. Laterization goes one step further and removes the silica as well. Its characteristics are: (1) the liberation of the silica from its various compounds; (2) the removal by solution of the lime and magnesia; (3) the oxidation of the ferrous to ferric iron; (4) the removal of the silica and the alkalies; (5) the concentration, as a residual mantle, of the alumina and ferric iron, with titania, chromic oxide, and other impurities; and (6) a sort of secondary dehydration leading to concretionary and pisolitic recemented masses, more or less abundantly disseminated through the mantle.

With this process in mind, the serpentine may be readily recognized as the parent of the iron-ore. Lime, magnesia, silica, and the alkalies have been largely if not wholly removed, and the iron and alumina have been concentrated. There is seven times as much iron in the ore as in the serpentine, and eight and one-half times as much alumina. About the same ratio appears to hold with the chromium, nickel, and titanium, which are nearly equally persistent with the iron and alumina. In short, there is no need to appeal to a hypothetical foreign source for any of the elements constituting the ore, either in whole or in part. No supposition involving transportation of material is required. Everything is at hand, and the history of the ore, as residual material derived directly from its underlying rock, is complete.

Reference has been made to the laterites of India. These laterites, being derived out of a different parent-rock from that in Cuba, naturally result in different end-products. Two types are recognized in India: the "high level" and the "low level." The former are *in situ*, while the latter are detrital, and differ from the usual detrital rocks only in the amount of lateritic cement. They have furnished the sources of iron-ore for numerous small native iron-smelting centers, and in this character they suggest an analogy with the Moa ores. As a matter of

fact, however, they are transported materials, and their iron-content, as found to-day, is undoubtedly due to mechanical sorting and concentrating, as well as, very possibly, to a certain amount of secondary chemical concentration. The presence of large amounts of mechanically-intermingled quartz still further distinguishes this type of lateritic ores from those at Moa. The true analogy lies between the Moa iron-ores and the Indian "high level" laterites, or (as they have within recent years in many cases proved to be) bauxites. The bauxites are found in the form of residual mantles, overlying their parent-rocks in a manner exactly similar to the occurrence of the Moa ores. The parent-rock, however, is dolerite and not serpentine. The result of laterization is therefore an end-product consisting chiefly of alumina, with ferric iron and other persistent impurities in proportion to the content of the original rock. Secondary dehydration marks the final stage of the process, as with the Moa ores, the result being a recemented pisolitic and nodular, frequently vesicular, reddish-brown material, resembling very closely in its appearance the recemented brown ores of Moa and other Cuban deposits. In short, whereas the parent and the offspring in the two cases respectively differ from one another, the process of generation is undoubtedly closely similar.

Dr. H. Warth[5] has presented a very interesting pair of analyses, one of dolerite and the second of bauxite, directly derived therefrom; both samples are from the western Ghats, near Bombay, India. These analyses are reproduced in Table III. for the purpose of comparison with the analyses of the Moa serpentine and iron-ore, given in Table II.

TABLE III.—*Analyses of Dolerite and Bauxite, India.*

	Dolerite. Per Cent.	Bauxite. Per Cent.
SiO_2	50.4	0.7
TiO_2	0.9	0.4
Al_2O_3	22.2	50.5
Fe_2O_3	9.9	23.5
FeO	3.6	......
MgO	1.5	......
CaO	8.4	......
K_2O	1.8	......
Na_2O	0.9	......
H_2O	0.9	25.0
	100.5	100.1

[5] *Geological Magazine*, decade V., vol. ii. (January, 1905).

The disappearance of the silica, lime, magnesia, and the alkalies, and the concentration of the alumina and iron, are here beautifully exemplified. There is 2.25 times as much alumina and 1.70 times as much iron in the bauxite as in the dolerite; thus a part of the iron has been removed, as was the case with the Moa ore. The titania also appears to have suffered partial removal. The solution of the silica is wonderfully complete.

Enough has been said to demonstrate that the Moa type of Cuban ore has its genesis through sub-aërial decay directly out of the underlying rock. The decay takes that form which has come to be known as laterization, a form peculiar to humid tropical climates, which has as its essential characteristic the breaking-up of the silica compounds and the more or less complete removal of the silica. The process has come nearest to completion at Moa, and that name may, therefore, be properly adopted to designate the type. The ores of other localities, Mayari, Cubitas, Taco, and Navas, are merely less-perfect examples of the same process.

The strange fact that laterization is confined to humid tropical climates has been the subject of more or less speculation. Sir Thomas Holland, Director of the Geological Survey of India, in his paper, On the Constitution, Origin, and Dehydration of Laterite,[6] has proposed a very ingenious and interesting theory in this connection. It should be remembered that he is speaking of lateritic bauxites which have been derived from dolerites, and, therefore, the reference is to aluminous silicates. It does not seem improper, however, to read magnesian for aluminous; in other words, the theory appears as applicable to the breaking-down of serpentine as of dolerite.

Sir Thomas Holland says:

> "To account for the fact that an aluminous silicate undergoes a more complete disintegration under tropical conditions than under the deep-seated and presumably high temperature conditions of kaolinization, the writer suggests that laterite is due to the agency of lowly organisms, possibly akin to the so-called nitrifying bacteria. With these are probably forms akin to the bacteria which oxidize and fix ferrous compounds, and which, precipitating the silica in the colloid form, permit its removal by the dilute alkaline solutions simultaneously formed."

We may now turn from the question of genesis to a consideration of the probable age and history of the Moa type of

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[6] *Geological Magazine*, IV., vol. x., p. 359.

brown ore-deposits. In this connection, I wish to acknowledge my indebtedness to A. C. Spencer for suggestions taken not only from his publication but also from verbal discussions.

The Cubitas, Mayari, Navas, and Taco deposits all occupy plateaus. Mr. Spencer says of the Cubitas in his paper, Three Deposits of Iron Ore in Cuba:[7]

"Within an area measuring roughly 10 miles east and west and 4 miles north and south, there are several flat-topped mesas rising 300 to 400 feet above the general level of an almost featureless plain. . . . The ore deposits are all surface mantles covering the plateau-like mesas."

The actual elevation of the mesas above sea-level is not given. The following description of the Mayari deposit has also been published:[8]

"The ore body is on the summit of a gently rolling plateau, roughly 10 miles long and 4 miles wide, with its principal axis lying northeast and southwest. Its elevation is about 1600 ft. at the northwestern extremity, which is nearest to Nipe Bay, and it rises toward the southwest to an elevation of 2200 to 2300 ft., with one peak reaching to 2600 ft. and another to 3200 ft.

The Taco deposit occupies a partly-dissected plateau, approximately 4 miles long by 1.5 miles wide, with the longer axis NE–SW., at a general elevation of 2,100 ft. above sea-level. The Navas plateau, extending 2.5 miles N–S. and 1.25 miles wide, is from 1,600 to 1,800 ft. above sea-level.

At Moa the conditions are different. The ore-mantle, adjacent to tide-water and extending thence inland for a distance of from 3 to 5 miles, lies on a surface exhibiting no very pronounced relief, the average grade shorewards being about 250 ft. to the mile. The final disappearance of the ore-mantle inland is obviously due to the usual processes of surface-denudation. The underlying serpentine first appears in the stream-beds, which may be wholly in bed-rock, with the ore still persisting on the inter-stream divides. Thus, the upper edge of the ore-body presents a series of fingers persisting upwards along the crests of the divides till it finally disappears on the divides as well as in the stream-bottoms.

There can be little doubt that the Moa type of ores were originally formed on an ancient peneplain, and probably occu-

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[7] *Bulletin No. 340, U. S. Geological Survey*, p. 324 (1908).
[8] *Iron Age*, vol. lxxx., No. 7, p. 421 (Aug. 15, 1907).

pied at that time vastly greater areas than to-day. The present plateaus are remnants of the ancient peneplain, and owe their elevated position to a period of uplift, accompanied by a certain amount of warping, fracturing, and probably partial subsidence. The plateau-deposits underwent more or less simple vertical uplift. The Moa area was tilted and possibly fractured, subsiding once more to somewhere near its present position and level. The peneplain period may perhaps be referred to the Upper Oligocene, since at that time, according to Hayes, Vaughan, and Spencer,[9] nearly the whole island of Cuba was submerged, excepting portions along the north and south shores of Santiago Province. The next succeeding (Miocene) period was one of general uplift. Quoting from the report of the above:

"There was folding and uplift during this period, the elevation along the axial line being greater than at the sides. . . . One would infer that the central portion of the Province of Santiago was more highly elevated than the coastal portion, since Upper Oligocene limestones occur in the central portion of that province at considerably higher elevations than along either the north or the south coast."

This last statement probably explains the tilting of the Mayari plateau towards the north, and also the tilting and possibly the breaking-off and resulting partial subsidence of the Moa block.

The uplift was followed by a rejuvenation of the streams and renewed denudation activity. The ancient peneplain was dissected and cut back until there were left only the present remnants, still carrying their original mantles of ore; and even these are in a geological sense, still rapidly disappearing. The upper portion of the Moa block has been planed off, while the lower portion owes its preservation to the fact that it lies so near sea-level that the stream-gradients have practically disappeared, and their power to dissect and remove has been reduced to a minimum.

Thus, the original development of the Moa type of ores should probably be referred to pre-Miocene times. During and since the period of uplift, vast quantities of these ores have undoubtedly been mechanically removed and dissipated. At the same time there is no reason to suppose that laterization-processes have ceased; it is in fact probable that new ores are

[9] *Report on a Geological Reconnoissance of Cuba*, included in *Civil Report of Brig.-Gen. Leonard Wood, Military Governor of Cuba*, vol. i., pp. 31 to 34 (1901).

forming to-day wherever opportunity offers. Such opportunity may be regarded as at a minimum on the plateaus, where heavy mantles of material lying nearly horizontally effectively protect the underlying rock from the action of surface-waters. Wherever the stream-beds have dissected the plateaus, however, revealing fresh areas of serpentine and inducing new and lower curves in the adjacent ground-water levels, the growth of laterite must be proceeding as it did in former times. The presence of numerous sinks appears to furnish evidence to support this statement. Thus, the descent of the lateritic zone must be keeping pace with the degradation of the land-surface, except where this is so rapid that products of decay are removed as soon as formed.

We infer, then, that there are two limits to the accumulation of laterite. On the one hand, too great an opportunity, which has led to its development on an undisturbed peneplain up to a point where the underlying rock is pretty well protected—in this case the decay will no doubt continue to proceed downwards, but at a greatly reduced rate; and, on the other hand, too little opportunity, through its removal mechanically as fast as it forms. The first limit may have been the condition of affairs at the close of the Upper Oligocene, when huge mantles of laterite had accumulated, to about their maximum depth. The second limit is that of to-day, except where, locally, intermediate sets of conditions may exist. At such points the ore-beds are undoubtedly increasing. On the whole, however, since the beginning of Miocene times, the probabilities are that very much greater quantities of the lateritic ores have been destroyed than have been formed.

The Moa deposit differs from the others chemically in being a more perfect type of laterization, and, structurally, in not occupying a plateau. It would appear that the deposit was originally a portion of a great peneplain, upon which was accumulated a thick mantle of ore through processes of sub-aërial decay common to humid tropical climates. At the time of the general uplift the Moa block apparently broke off and partly subsided once more, tilting toward the north. Renewed denudation then planed off the upper edges of the block, removing the ore-mantle and cutting down into the underlying rock. Any further ore forming at the higher levels was removed as fast as

it formed. Lower down, however, transportation was at its minimum, and the ore already existing not only remained undisturbed, but probably increased somewhat in depth. Furthermore, through saturation by surface-waters, the process of laterization was advanced nearly to theoretical completeness, and at the lower levels, representing the heart of this great ore-field, there are occasionally found mantles of ore exceeding 60 ft. in thickness, containing a minimum of silica.

The structural history of this field may raise the question, why attribute the accumulations of ore entirely to decay in place? Was there not exceptional opportunity here for mechanical transport, or transport in solution with secondary concentration? The answer is, the absolute absence of evidence or appearance of such accumulation through transport of material. The ores as found are satisfactorily accounted for in their entirety by the decay of the underlying rock. The analyses of many samples, representing in a most thorough manner wide areas, are remarkably uniform, although many were taken from localities where opportunity for secondary concentration or accumulation would be much better than at other localities. In the absence of any evidence to the contrary, the conclusion must be that the ore-bearing materials which have been removed from the upper levels were simply dissipated by the waters removing them. They do not appear to have contributed to the ores of the lower levels, either by increasing the bulk of the accumulation or by any process of enrichment.

Similar questions of possible transport of material cannot be raised by the structural conditions attendant upon the other fields. Hence, the theory of development of ore in place through laterization-processes offers the only ready and entirely satisfactory explanation for the ore-deposits in those fields; and that the laterization has not been as perfect as at Moa may be due to any number of readily-suggested causes.

The development of these huge fields of iron-ore in Cuba has directed study upon several metallurgical problems attending their use in the manufacture of iron and steel, chiefly in connection with the high alumina- and chromium-contents. Exhaustive studies and experiments on Mayari ores have been carried out within the last few years by the Pennsylvania Steel Co., and it has been announced that all the difficulties

have been solved, and steel rails of more than usual excellence have been manufactured from these ores. It is not within the scope of this paper to go into metallurgical details. I merely refer to this feature and its happy solution in a congratulatory frame of mind. The cheapness with which the ore can be extracted must have presented itself to any one of mining experience. The high moisture-content, hygroscopic plus combined, together with the clayey structure, makes it very desirable to treat the ore before shipping by some agglomerating-process under fairly-high temperatures. With the former difficulties attending the presence of high alumina and high chromium removed, the agglomerated or nodulized ore, practically free from moisture and in that condition carrying more than 50 per cent. of metallic iron, will undoubtedly find a ready market on the Atlantic seaboard, and in all probability new plants will be erected on the seaboard to profit by this opportunity.

No. 33 SEPTEMBER 1909

Bulletin of the American Institute of Mining Engineers.

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EDITORIAL OFFICE AT 29 WEST 39TH STREET, NEW YORK, N. Y.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.

No. 33 SEPTEMBER 1909

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Single copies (including postage), $1 each; to members of the Institute, public libraries, etc., 50 cents each.

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SECTION I.—INSTITUTE ANNOUNCEMENTS.

This section contains announcements of general interest to the members of the Institute, but not always of sufficient permanent value to warrant republication in the volumes of the *Transactions*.

SECTION II.—TECHNICAL PAPERS AND DISCUSSIONS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

A detailed list of the papers contained in this section is given in the Table of Contents. They have been so printed and arranged (blank pages being left when necessary) that they can be separately removed for classified filing, or other independent use.

A small stock of separate pamphlets, duplicating the technical papers given in Section II. of this Bulletin, is reserved for those who desire extra copies of any single paper.

Comments or criticisms upon all papers given in this section, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

All communications concerning the contents of this Bulletin should be addressed to Dr. Joseph Struthers, Assistant Secretary and Editor, 29 W. 39th St., New York, N. Y. (Telephone number 4600 Bryant).

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

INSTITUTE ANNOUNCEMENTS.

Spokane Meeting and Excursions.

The XCVIIth meeting of the Institute for the reading and discussion of papers will be held at Spokane, Wash., beginning Monday, Sept. 27, 1909, as already announced in Special Circulars of May 8 and Aug. 14, 1909, and in the *Bulletins* for June, July, and August.

The meeting will be preceded by a tour through Yellowstone Park and visits to Butte and Anaconda, and will be followed by visits to the Cœur d'Alene district, to the Alaska-Yukon-Pacific Exposition at Seattle, to the mining- and smelting-plants near Salt Lake, and to the iron- and lead- and zinc-plants at Pueblo. The special train for the party will leave Chicago, September 16, and return about October 12. Details of the itinerary are given later in this announcement.

The Secretary of the Spokane Local Committee is Lyndon K. Armstrong, 615 Hyde Block, Spokane, Wash., to whom should be addressed all inquiries concerning local matters. The control of the papers and proceedings will remain in charge of the Institute office, as usual.

It is greatly to be desired, both in behalf of the Institute and in recognition of the many courtesies to be extended on the trip, that the excursion party be made larger. Exceptional facilities to visit the many mines and smelters have been freely provided; and members are requested to make all reasonable effort to attend the meeting at Spokane and participate in the excursions.

For those who cannot join in the entire trip, it may be possible to provide special Pullman accommodations after leaving Yellowstone Park. In order to make the necessary arrangement, however, the names of members and guests contemplating this partial trip should be sent at once to Theodore Dwight, 29 West 39th Street, New York City.

The general cost of the trip per member will be $300, which includes transportation, berth, and meals for the entire trip from

Chicago back to Chicago, almost 6,000 miles, and the tour through Yellowstone Park, occupying in all about 30 days. Special accommodations on the train will be furnished at the following rates:

Chicago to Chicago, including transportation, berth, and meals,	$300.00
State-room for one person, $200.00 extra, or	500.00
State-room for two persons, total,	700.00
Drawing-room for two persons, total,	800.00
Drawing-room for three, total,	1,100.00

Full particulars of the sessions, entertainments, and excursions in and around Spokane will be given in the program of the Local Committee, to be furnished on registration at Institute Headquarters in Spokane. The first session will be held on Monday evening, September 27. It is not now practicable to give all details of the papers and discussions to be presented; but the list so far as completed contains many papers of professional importance to mining engineers, economic geologists, metallurgists, and chemists.

The following list, comprising the titles of papers already announced for the Spokane meeting, will doubtless be increased until the time of the sessions, and, while there is no definite time-limit for the receipt of additional papers, authors intending to be present and desiring to have time assigned to them, are requested immediately to notify Dr. Struthers in writing. In the absence of such prompt notice, the desired assignment of time cannot be definitely promised.

The majority of the papers in the list will be, as usual, read by title, or presented in brief oral abstract only, and printed copies of many of them will be on hand at the sessions.

Members desiring to see any of these papers for the purpose of discussion at the meeting, are requested to communicate with Dr. Struthers, who will forward in reply advanced copies of the desired papers, if they are in print, or will give opportunity during the meeting for an inspection of the manuscripts.

1. Dust-Explosions in Coal-Mines. By Franklin Bache, Fort Worth, Ark.
2. The Formation and Enrichment of Ore-Bearing Veins. By George J. Bancroft, Denver, Colo.
3. Modern Practice of Ore-Sampling. By D. W. Brunton, Denver, Colo.

4. Need of Instrumental Surveying in Practical Geology. By Benjamin Smith Lyman, Philadelphia, Pa.
5. Modern Progress in Mining and Metallurgy in the Western United States. By D. W. Brunton, Denver, Colo.
6. The Assay and Valuation of Gold-Bullion. By Frederic P. Dewey, Washington, D. C.
7. The Cyanidation of Silver-Ores in Mexico. By Albert F. J. Bordeaux, Thonon les Bains, France.
8. The Fushun Colliery, South Manchuria. By Warden A. Moller, Tientsin, China.
9. Protective Value of Humidity in Dusty or Gaseous Coal-Mines. By James Ashworth, Congleton, England.
10. A Study of the Ventilating-System of the Comstock Mines. By George J. Young, Reno, Nev.
11. Professional Ethics. By Victor G. Hills, Denver, Colo.
12. An Adjustable Pyrometer-Stand. By L. W. Bahmey, Palo Alto, Cal.
13. The Ruble Hydraulic Elevator. By J. McD. Porter, Spokane, Wash.
14. Notes on Construction of a Glass Mine-Model. By Edmund D. North, Tonopah, Nev.
15. Postscript to paper, The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes. By H. O. Hofman and W. Mostowitsch, Boston, Mass.
16. Cyaniding Slime. By Mark R. Lamb, Milwaukee, Wis.
17. Influence of Ingot-Size on the Degree of Segregation in Steel Ingots. By Henry M. Howe, New York, N. Y.
18. The Filing of Assay-Samples for Use in Technical Laboratories. By Louis D. Huntoon, New Haven, Conn.
19. Dredging for Gold in French Guiana. By Albert F. J. Bordeaux, Thonon les Bains, France.
20. Mining Industry of Nicaragua. By T. Lane Carter, Bluefields, Nicaragua, Central America.
21. Barite Industry of the United States. By A. A. Steel, Fayetteville, Ark.
22. Federal Coal-Mines in the Philippines. By Oscar H. Reinholt, Pasadena, Cal.
23. Discussion of the paper of Charles R. Keyes, Genesis of the Lake Valley, N. M., Silver-Deposits. By William M. Courtis, Detroit, Mich.
24. Discussion of the paper of Henry M. Howe, Piping and Segregation in Steel Ingots. By P. H. Dudley, New York, N. Y.
25. Discussion of the paper of Charles R. Keyes, Ozark Lead- and Zinc-Deposits; Their Genesis, Localization, and Migration. By E. R. Buckley, Flat River, Mo.
26. Discussion of the paper of D. F. Hewett, Vanadium-Deposits in Peru. By James F. Kemp, New York, N. Y.
27. Discussion of the paper of Hofman and Hayward, Pan-Amalgamation; an Instructive Laboratory-Experiment. By E. A. H.t Tays, San Blas, Sinaloa, Mexico.
28. The Influence of Bismuth on Wire-Bar Copper. By H. N. Lawrie, Portland, Oregon.
29. A Study of the Critical Condition in Appalachian Coal-Mines Affecting Explosions in which Coal Dust is a Factor. By N. H. Mannakee, Williamson, W. Va.

30. Influence of Top-Lag on the Depth of the Pipe in Steel Ingots. By Henry M. Howe, New York, N. Y.
31. The Combustion-Temperature of Carbon and Its Relation to Blast-Furnace Phenomena. By C. P. Linville, State College, Pa.

The following list comprises the names of members and guests who have reserved accommodations for the entire trip up to Sept. 10, 1909:

Mr. and Mrs. W. S. Ayres.
Mr. F. H. Bostwick.
Mr. and Mrs. Owen Brooke.
Miss Brooke.
Mr. D. W. Brunton.
Mr. Charles Catlett.
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Prof. and Mrs. Wm. Kent.
Mr. A. W. Lawton.
Mr. and Mrs. John Lilly.
Mr. William Lilly.
Mr. and Mrs. W. S. Mitchell.
Mr. and Mrs. William Nesmith.
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Dr. R. W. Raymond and Alfred R. Bellinger.
Mr. W. L. Saunders.
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Mr. W. O. Snelling.
Mr. George Steiger.
Dr. Joseph Struthers.
Mr. and Mrs. A. E. Vaughan.
Mr. S. T. Wellman.
Mr. C. H. Weiss.

Itinerary.

The following condensed itinerary is provisionally announced, subject to such changes and additions as may be made by the Local Committees or by Mr. Dwight, in charge of the railroad arrangements.

Thursday,	Sept. 16th	Lv. Chicago..........9.00 A. M.	via C. M. & St. P.
		Arr. St. Paul..........9.40 P. M.	via C. M. & St. P.
		Lv. St. Paul about 10.00 P. M.	via N. P. R. R.
Friday,	" 17th	En Route.	
Saturday, to Thursday,	" 18th " 23d	In Yellowstone Park. Lv. Night	via N. P. R. R.
Friday,	" 24th	Butte. Ar. A. M. Lv. Night.	
Saturday,	" 25th	Anaconda. Ar. A. M. Lv. Night.	
Sunday, to Thursday,	" 26th " 30th	In Spokane. Lv. A. M.	via N. P. R. R.
Thursday, to Sunday,	" 30th Oct. 3d	In Seattle. Lv. Night.	

Monday,	Oct.	4th	.........Tacoma. Ar. A. M. Lv. Night.
Tuesday,	"	5th	.........Portland. Lv. A. M. via O. R. & N. Co.
Wednesday, to Saturday,	" "	6th } 9th }	......In Salt Lake. Lv. 8 P. M. via D. & R. G.
Sunday,	"	10th	.........Glenwood Springs. Ar. A. M. Lv. Night.
Monday,	"	11th	.........Pueblo. Ar. A. M. Lv. 4.30 P. M. via C. R. I. & P.
Tuesday,	"	12th	.........Arr. Chicago. Night.

Meetings of Other Societies.

American Society of Mechanical Engineers.—The American Society of Mechanical Engineers will hold monthly meetings in the Engineering Societies Building, New York, on the evenings of October 12 and November 9. The annual meeting will be held in New York December 7–10. Meetings of the Society are also to be held throughout the fall and winter in St. Louis and Boston.

In connection with the Hudson-Fulton celebration in September and October in New York, and as a contribution on the part of the engineering profession, there will be placed in the rooms of the American Society of Mechanical Engineers a valuable exhibit of objects of interest relating to the early history of steam-navigation. The exhibit will include the following objects belonging to the Society: an oil-portrait of Robert Fulton by himself; original drawings by Fulton; a table that belonged to Fulton; reproduction in bronze of the Fulton drawing of the steamer *Potomac*, 1820; letters of Fulton's workmen concerning the first steamboat; portraits of prominent marine engineers, etc. Steps have also been taken to secure for the exhibit a collection of Ericsson models from the United Engineering Society, and models of early steamboats, including Fulton's *Clermont*, from the Smithsonian Institution.

Conservation of Mineral Resources.

The papers on the Conservation of Mineral Resources, by members of the U. S. Geological Survey, which were included in the Report of the National Conservation Commission, have been reprinted in *Bulletin No.* 394 *of the U. S. Geological Survey,*

entitled, *Papers on the Conservation of Mineral Resources.* These papers are:

Coal-Fields of the United States. By M. R. Campbell and E. W. Parker.
Estimates of Future Coal Production. By Henry Gannett.
The Petroleum Resources of the United States. By D. T. Day.
Natural-Gas Resources of the United States. By D. T. Day.
Peat Resources of the United States, Exclusive of Alaska. By C. A. Davis.
Iron-Ores of the United States. By C. W. Hayes.
Resources of the United States in Gold, Silver, Copper, Lead, and Zinc. By Waldemar Lindgren.
The Phosphate Deposits of the United States. By F. B. Van Horn.
Mineral Resources of Alaska. By A. H. Brooks.

A large edition of this *Bulletin* has been printed, and copies may be obtained without charge on application to the Director of the U. S. Geological Survey, Washington, D. C.

Office Facilities for Visiting Members.

A separate room in the suite occupied by the American Institute of Mining Engineers on the ninth floor of the United Engineering Society Building, has been equipped with furniture and telephone extension for the temporary use of members of the Institute or of sister societies, or visitors suitably accredited.

Members of the Institute visiting New York for a short time, who need office facilities during their stay, or members residing in the city who need temporary office accommodation, can arrange to have set apart for their exclusive use a room, equipped with office furniture, telephone, etc., in the suite of the Institute. It is not the intention to give possession of the room to any individual for an indefinite time, but to offer to members of the Institute an opportunity to acquire a well-located, well-equipped business headquarters to carry on transactions which would not warrant the establishment of a permanent office. The room devoted to this purpose is entirely separate from the reception- and writing-rooms for the general use of the members. A small fee will be required for the use of the facilities furnished. For the conditions of this privilege, inquiry should be made at the office of the Secretary of the Institute.

How to Use the "Transactions" of the Institute.

Buy a copy of the Complete Analytical and Alphabetical Index of Volumes I. to XXXV., inclusive.

If you own a full set of the *Transactions*, this Index will make the whole of it instantaneously available without detailed research into each volume separately.

If you do not own such a set, this Index will be even more valuable, for it will show you what particular papers you need to know more about, and perhaps to study. Thus, any person possessing this Index can ascertain at once what has been published in the *Transactions* on a given question, and can learn, by writing to the Secretary, what is its nature, whether it is still to be had in pamphlet form, where it can be consulted in a public library, at what cost it can be copied by hand, etc., etc.

In short, to those who own complete sets of the *Transactions*, this Index will be a great convenience; but to those who do not, it will be a professional necessity.

This volume is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. It has not been stereotyped, and the edition is limited to 1,600 copies. The price of the volume, bound in cloth, is $5, and bound in half-morocco to match the *Transactions*, $6. The delivery charges will be paid by the Institute on receipt of the above price.

Hydrographic Chart.

Owing to the great value to hydrographers of the chart contained in the paper, A Graphic Solution of Kutter's Formula, by L. I. Hewes and Joseph W. Roe (*Bulletin No.* 29, May, 1909, p. 454), a special edition for office or field use has been printed on durable cloth. Copies of this separate chart may be obtained, at a cost of 50 cents each, on application to the office of the Secretary of the Institute.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

RULES.

For the protection and convenience of members, the following rules have been adopted:

The Secretary of each Society will, upon application, issue to any member of his Society in good standing a personal, non-transferable card, entitling him to the use of the Libraries in the alcoves of the Reading-Room.

This card, as well as any card of introduction given to a non-member, must be signed by the person receiving it, and surrendered at the desk at the time of its presentation. At every visit he must identify himself by signing his name in the registry.

Strangers who desire to enjoy the privilege of entering the alcoves are requested to present either letters of introduction from members, or cards, such as will be furnished upon application by the Secretary of each Society. The first two alcoves are free to all; and admission to the inside alcoves is given upon proper introduction.

The above rules apply to all persons except officers of the three Societies, personally known as such to the librarians.

The librarians are not permitted to lend to any person any catalogued pamphlet or volume, unless authorized in writing so to do by the Secretary or Chairman of the Library Committee of the Society to which the pamphlet or volume belongs.

Any person discovering a mutilation or defect in any book of the libraries is requested to report it to the librarian on duty.

Library Additions.

From Aug. 1 to Sept. 1, 1909.

DIE ADAMELLOGRUPPE. By Wilhelm Salomon. Wien, K. K. Geologischen Reichsanstalt, 1908. (Exchange.)

AMERICAN POLYTECHNIC JOURNAL. Vol. I., January-June, 1853. Washington, 1853. (Gift of Dr. Leonard Waldo.)

BUCKS COUNTY HISTORICAL SOCIETY. Collection of Papers Read. Vol. 3. Easton, Chemical Publishing Co., 1909. (Gift of B. F. Fackenthal, Jr.)

[SECRETARY'S NOTE.—This completes the series of which Vols. I. and II. have been already noticed in the *Bulletin*. The final volume is perhaps more, certainly not less, interesting than its predecessors. Bucks county is specially rich in historical associations in three departments, connected respectively with the Lenni Lenape tribe (made famous by the "Leatherstocking" tales of Fenimore Cooper); the Moravians, Mennonites, and so-called "Pennsylvania Dutch" settlers; and the campaigns of Washington during the most critical period of the American Revolution. All these, together with the histories of many old families and illustrious citizens, are represented in this volume, which contains also an interesting illustrated description of the phonolite outcrops or "ringing rocks" of that region, from the pen of Mr. Fackenthal himself.—R. W. R.]

CARNEGIE INSTITUTE OF PITTSBURGH. 13th Celebration of Founder's Day, April 29, 1909. Pittsburgh, 1909. (Gift.)

CARNEGIE LIBRARY OF PITTSBURGH. Annual Report to the Board of Trustees, 13th. Pittsburgh, 1909. (Exchange.)

CARNEGIE MUSEUM OF PITTSBURGH. Annual Report of the Director, 12th. Pittsburgh, 1909. (Gift.)

CHAMBER OF COMMERCE OF PITTSBURGH. Annual Report of President L. S. Smith, May 13, 1909. (Gift.)

CHARACTERISTICS OF THE TURBINE PUMP. By F. Ray. N. p., n. d. (Gift of Alberger Pump Co.)

COAL MINING. Pts. 1–3. By D. Burns and G. L. Kerr. London, Whittaker & Co., 1907–08. (Purchase.)

COMMISSION D'ENQUETE SUR LA DURÉE DU TRAVAIL DANS LES MINES DE HOUILLE. Statistique des Accidents Miniers. Par L. Delruelle et A. Delmer. Bruxelles, 1909. (Gift.)

COMPARATIVE STATISTICS OF LEAD, COPPER, SPELTER, TIN, ALUMINIUM, NICKEL, QUICKSILVER, AND SILVER. Compiled by the Metallurgischen Gesellschaft A.-G. 15th Annual Issue, 1899–1908. Frankfort-on-the-Main, 1909. (Gift.)

CONSULAR FEES AND INVOICES OF LATIN AMERICAN COUNTRIES. Practical Information for Exporters to Latin America. Washington. 1909. (Exchange.)

CONTRIBUTIONS TO ECONOMIC GEOLOGY, 1908. Pt. I.—Metals and Non-metals, Except Fuels. (Bulletin No. 380, U. S. Geological Survey.) By C. W. Hayes and W. Lindgren. Washington, U. S. Government, 1909. (Exchange.)

CRISTALLISATIONS DES GROTTES DE BELGIQUE. By W. Prinz. Bruxelles, 1908. (Exchange.)

DICTIONARY OF CHEMICAL AND METALLURGICAL MATERIAL. New York, 1909. (Gift of *Electrochemical and Metallurgical Industry*.)

EFFECT OF OXYGEN IN COAL. (Bulletin No. 382, U. S. Geological Survey.) By David White. Washington, U. S. Government, 1909. (Exchange.)

THE ENGINEER AND THE STATE. Presidential Address of A. C. Lane to the Michigan Engineering Society, Feb. 13, 1909. N. p., n. d. (Gift.)

ENGINEERS' CLUB OF PHILADELPHIA. Directory, 1909. Philadelphia, 1909. (Exchange.)

ETHERIDGE GOLDFIELD (2d report). By W. E. Cameron. (Publication No. 219, Queensland Geological Survey.) Brisbane, 1909. (Exchange.)

FAUNA OF THE CANEY SHALE OF OKLAHOMA. (Bulletin No. 377, U. S. Geological Survey.) By G. H. Girty. Washington, U. S. Government, 1909. (Exchange.)

FIGURE OF THE EARTH AND ISOSTASY FROM MEASUREMENTS IN THE UNITED STATES. By J. F. Hayford. Washington, U. S. Government, 1909. (Exchange.)

GAS PRODUCERS AND GAS FIRING. By Ernest Schmatolla. London, 1909. (Gift of Author.)

GEOLOGICAL RECONNAISSANCE IN NORTHERN IDAHO AND NORTHWESTERN MONTANA. (Bulletin No. 384, U. S. Geological Survey.) By F. C. Calkins. Washington, U. S. Government, 1909. (Exchange.)

GEOLOGICAL REPORT UPON THE GOLD AND COPPER DEPOSITS OF THE PHILLIPS RIVER, GOLDFIELD. (Bulletin No. 35, Western Australia Geological Survey.) By H. P. Woodward. Perth, 1909. (Exchange.)

GENETIC RELATIONS OF SOME GRANITIC DIKES. By A. C. Lane. (Reprint.) N. p., n. d. (Gift.)

GUNN'S PLAINS, ALMA, AND OTHER MINING FIELDS, NORTHWEST COAST. (Bulletin No. 5, Tasmania Geological Survey.) By W. H. Twelvetrees. Hobart, 1909. (Exchange.)

IGNEOUS ROCKS. Vol. I. By J. P. Iddings. New York, J. Wiley & Sons, 1909. Price, $5. (Gift of Publishers.)

[SECRETARY'S NOTE.—This volume deals with the composition, texture, and classification of the igneous rocks. The second volume will be descriptive, local occurrences, etc. In the *Engineering and Mining Journal*, I have expressed at some length my opinion of the book. It is a masterly treatise in the newest field of geological science, and worthily represents the progress made in that field by American as well as foreign investigators. The most characteristic and interesting part of the volume is the chapter on the famous Quantitative System of Classification, for which Dr. Whitman Cross, Prof. Louis V. Pirsson, and Dr. Henry Washington are jointly responsible with Mr. Iddings.—R. W. R.]

ILLINOIS—BUREAU OF LABOR STATISTICS. Annual Report of the Illinois Free Employment Offices, 10th, 1908. Springfield, 1909. (Exchange.)

INSTITUTION OF MINING AND METALLURGY. Bulletin No. 59. London, 1909. (Exchange.)

IRON AND STEEL INSTITUTE. Journal, No. I., 1909. London, 1909. (Exchange.)

——— Rules and List of Members Corrected to July 28, 1909. (Exchange.)

LATIN AMERICA THE LAND OF OPPORTUNITY. Washington, 1909. (Exchange.)

DER MAGNESIT. By Robert Scherer. Wien und Leipzig, 1908. (Purchase.)

THE MAN WITH THE PICK VS. THE MAN WITH THE BOOK. St. Louis, n. d. (Gift of Practical Miner Co.)

MESSINGWERK. By P. G. Gurnik. Wien und Leipzig, 1908. (Purchase.)

METAL CORROSION AND PROTECTION. References to Books and Magazine Articles. Ed. 2. Pittsburgh, Carnegie Library, 1909. (Exchange.)

MICHIGAN GEOLOGICAL SURVEY. Report, 1907. Lansing, 1908. (Exchange.)

MINERAL INDUSTRY. Vol. 17. New York, McGraw-Hill Book Co., 1909. (Gift of Publishers.)

[SECRETARY'S NOTE.—In the *Engineering and Mining Journal* of Aug. 21, 1909, I have stated at some length my opinion of this volume, as a supplement and climax to its sixteen predecessors. The grounds of that opinion were mainly: (1) the symposium of contributions from representative officers of practically all the State geological surveys, of which there are now nearly 40 (including mining bureaus, etc., constituting substantial equivalents, so far as economic geology is concerned); (2) the wide representation of American technical schools, through contributions from distinguished instructors; (3) the articles of eminent practicing mining engineers and metallurgists; and (4) the work of Mr. Ingalls himself and his able office-staff, evidenced in many articles, but chiefly, perhaps, in the early appearance of the book itself, the editorial work upon which was finished at the end of May, and the last proof-reading in June.—R. W. R.]

MINING INSTITUTE OF SCOTLAND. List of Members, 1908–1909. Hamilton, 1909. (Exchange.)

MONOGRAPHIC REVISION OF THE COLEOPTERA BELONGING TO THE TENEBRIONIDE TRIBE ELEODIINI, INHABITING THE UNITED STATES, LOWER CALIFORNIA, AND ADJACENT ISLANDS. (Smithsonian Institution. U. S. National Museum Bulletin No. 63. By F. E. Blaisdell. Washington, 1909. (Exchange.)

NEW ORLEANS SEWERAGE AND WATER BOARD. Semi-Annual Report, 18th. New Orleans, 1908. (Gift.)

NORTHWEST MINING NEWS. Vol. 1, No. 4–Vol. 5, No. 2. Spokane, 1907–1909. (Gift of *Northwest Mining News.*)

NOTES ON EXPLOSIVE MINE GASES AND DUSTS, WITH SPECIAL REFERENCE TO EXPLOSIONS IN THE MONONGAH, DARR, AND NAOMI COAL-MINES. (Bulletin No. 383, U. S. Geological Survey.) By R. T. Chamberlin. Washington, U. S. Government, 1909. (Exchange.)

NOUVEAU MANUEL COMPLET DU FONDEUR. By A. Gillot and L. Lockert. Vols. 1-2. Paris, 1905. (Purchase.)

POLYTECHNIC INSTITUTE OF BROOKLYN. Catalogue of the College of Engineering, 1909–1910. (Gift.)

POUSSIERES DE HOUILLE. By M. J. Daniel. Paris, Dunod et E. Pinat, 1909. (Gift of Publishers.)

PRE-CAMBRIAN GEOLOGY OF NORTH AMERICA. (Bulletin No. 360, U. S. Geological Survey.) By C. R. Van Hise and C. K. Leith. Washington, U. S. Government, 1909. (Exchange.)

QUEBEC—COLONIZATION, MINES, AND FISHERIES DEPARTMENT. Mining Operations in the Province, 1908. Quebec, 1908. (Exchange.)

REAR CAR FOR WOMEN ON SUBWAY TRAINS. Decision Dismissing Complaint, Aug. 3, 1909. New York, 1909. (Gift of Public Service Commission for the First District, State of New York.)

REINFORCED CONCRETE IN EUROPE.—By A. L. Colby. Easton, Chemical Publishing Co., 1909. Price, $3.50. (Gift of Author.)

[SECRETARY'S NOTE.—This book contains a summary of foreign practice, from the pen of an eminent and acknowledged international authority, in a department of constructive engineering which has recently acquired great importance, and presages already a revolution in the engineering materials and methods of the field to which it belongs. Incidentally, the immense and growing use of concrete has an important bearing upon the burning question of the "conservation of natu-

ral resources." Whatever prophecies of disaster may have been made concerning the exhaustion of our timber or our iron-ore, nobody has yet ventured to limit our supply of the materials for concrete. Information as to that material from a competent expert like Mr. Colby will be welcome to all constructing engineers.—R. W. R.]

RENSSELAER POLYTECHNIC INSTITUTE. Formal Opening of the Russell Sage Laboratory, June, 1909. Troy, 1909. (Gift.)

REPORT ON THE TUNGSTEN ORES OF CANADA. By T. L. Walker. Ottawa, 1909. (Exchange.)

RESULTS OF MAGNETIC OBSERVATIONS MADE BY THE COAST AND GEODETIC SURVEY BETWEEN JULY 1, 1907, AND JUNE 30, 1908. By R. L. Faris. Washington, U. S. Government, 1909. (Exchange.)

SCHAEBERLE, BECKER AND THE COOLING EARTH. (Reprint.) N. p., n. d. (Gift.)

SHELBY IRON CO. Report, 1909. Birmingham, 1909. (Gift of Shelby Iron Co.)

SOCIÉTÉ GÉOLOGIQUE DU NORD. Annales 36, 1907. Lille, 1907. (Exchange.)

SOCIETY FOR THE PROMOTION OF ENGINEERING EDUCATION. Proceedings. Vol. 16. Brooklyn, 1909. (Gift of Society.)

STATISTIQUE DES ACCIDENTS MINIERS. By L. Delruelle and A. Delmer. Bruxelles, 1909. (Gift of Dr. Struthers.)

STRUCTURAL MATERIALS IN PARTS OF OREGON AND WASHINGTON. (Bulletin No. 387, U. S. Geological Survey.) By N. H. Darton. Washington, U. S. Government, 1909. (Exchange.)

TIMBERING AND MINING. By W. H. Storms. New York, McGraw-Hill Book Co., 1909. Price, $2 net. (Gift of Publishers.)

[SECRETARY'S NOTE.—It is noteworthy that the department of mine-timbering has received so little attention in American technical literature. Probably the reason is, that our mining-operations have been so largely conducted under the conditions of abundant timber-supply, high wages for labor, high interest on capital, and so large a value of mine-products as to warrant relative waste of timber for compensating gain in the rate of immediate extraction. Valuable general rules and systems of scientific economy in a given feature of practice cannot be developed until economic pressure has been experienced in that particular feature. Thus it came to pass that the mining engineers who exploited the great bonanzas of the Comstock lode impressed upon American practice for many years a system of timbering which, though adapted to their immediate needs, was, for general conditions and for future decades, perhaps the most wasteful and the most perilous that could be devised. "Caves," "crushes," and mine-fires innumerable have proved this proposition. But, according to the familiar proverb of the Pacific coast, "a rich mine makes a good manager;" that is to say, the methods of a manager who, with a margin of profit large enough to cover all mistakes, drives hard, extracts ore rapidly, and enables his company to pay large and frequent dividends—no matter at what cost—redound to his professional credit, and are deemed worthy of imitation. And hence the Comstock methods of mining were everywhere imitated, often with disastrous results.

Meanwhile, however, the best way to use timber, if not the best way to avoid its excessive use, had to be learned by practical miners, even for small mines; and American literature was singularly deficient in manuals of this subject.

In 1873, Mr. Storms, the author of the book before me, and a veteran employee of the California State Mining Bureau, published as *Bulletin No.* 2 of that department a little treatise on Methods of Mine-Timbering, which was, I believe, the

first American monograph on this theme. Successive editions of it were eagerly absorbed by the miners of the West, and it was republished abroad in English, French, and German, as a valuable original contribution to technical literature. It is now out of print, and second-hand copies, even, are scarce. But this is not to be lamented; for, notwithstanding its great immediate value, Mr. Storms's treatise was a discussion rather of the use of timber in mines than of the ways of diminishing that use. In other words, it was influenced by the natural American notion that timber was the one thing that need not be spared in mining, if labor and time could be saved and immediate safety secured by its unlimited employment.

Fortunately, the author has lived long enough to see a great change in the economic conditions of mining, and to publish a new treatise, which, while retaining the valuable results of his earlier knowledge of American practice, takes into consideration the newer methods of saving timber as well as of using it. This extension involves the discussion of mining-methods, outside of the timbering involved; for, of course, in showing how timber may be, to any extent, dispensed with, the author is obliged to describe the methods and materials which make that result feasible. The result is a simple, clear, and practical discussion of a good deal outside of mere timbering. In short, Mr. Storms has given us a highly useful hand-book covering a large part of the art of mining, and containing numerous suggestions, valuable not only to the uneducated miner, but also to the professional engineer. In fact, it is the professional engineer more than anybody else who welcomes and appreciates those results and traditions of practice which he may easily have missed in his study of theory.—R. W. R.]

U. S.—CENSUS BUREAU. 13th Census, taken in 1910. Supervisors' Districts. (U. S. Census, Bulletin No. 98.) Washington, 1909. (Exchange.)

UNIVERSITY OF WISCONSIN. Catalogue, 1908–1909. (Exchange.)

WESTERN AUSTRALIA—GEOLOGICAL SURVEY. Annual Progress Report, 1908. Perth, 1909. (Exchange.)

GIFT OF HILL PUBLISHING COMPANY.

ANNALES DES MINES DE BELGIQUE. Vol. 13, No. 4. Bruxelles, 1908.

AUSTRALIA—COMMONWEALTH BUREAU OF CENSUS AND STATISTICS. Trade and Customs and Excise Revenue, 1906. Pt. I. Melbourne, 1907.

——— Trade, Shipping, Oversea Migration, and Finance. (Bulletin Nos. 21–23. Melbourne, n. d.

AUSTRIA—K. K. MINISTERIUM FÜR OFFENTLICHE ARBEITEN. Statistik des Bergbaues in Osterreich, 1907. Zweite lieferung. Wien, 1908.

——— K. K. ACKERBAUMINISTERIUMS. Statistiches jahrbuch. Dritte lieferung. Wien, 1907.

BELGIUM—MINISTÈRE DE L'INDUSTRIE ET DU TRAVAIL. Statistique des Industries Extractives et mètallurgiques et des Appareils a Vapeur, 1907. Bruxelles, 1908.

——— COMMISSION D'ENQUÊTE SUR LA DURÉE DU TRAVAIL DANS LES MINES DE HOUILLE. Statistiques Recueillies par l'Administration des Mines. Bruxelles, 1908.

BRITISH INDIA. Production of Minerals in British India and Native States During 1907. N. p., n. d.

BROWN, H. Y. L. Record of the Mines of South Australia. Ed. 4. Adelaide, 1908. (2 copies.)

CANADA YEAR BOOK, 1908. 2d series. Ottawa, 1909.

CANADA—CUSTOMS COMMISSION. Tables of Trade and Navigation, 1908. Ottawa, 1908.

CHILE. Estadistica Commercial de la Republica de Chile, 1904, 1906. Valparaiso, 1905, 1907.

GREAT BRITAIN. Mines and Quarries. General Report and Statistics. 1904, Pt. IV.; 1905, Pt. IV.; 1906, Pts. III., IV.; 1907, Pts. I., IV. London, 1906–1909.

GREECE. Commerce Spècial de la Statistique, 1906. Athènes, 1908.

HANDELSSTATISTISCHEN DIENSTE DES K. K. HANDELSMINISTERIUMS. Statistik des Auswärtigen Handels, 1907. Vol. I. Wien, 1908.

——— Statistische Ubersichten Betreffend den Auswärtigen Handel. Pt. 12, Dec., 1908. Wien, 1909.

HOLLAND, T. H. Sketch of the Mineral Resources of India. Calcutta, 1908.

INDIA—GEOLOGICAL SURVEY. Records. Vol. 36, Pt. 2; Vol. 37, Pt. 1. Calcutta, 1907, 1908.

——— MINES DEPARTMENT. Report of the Chief Inspector of Mines, 1906, 1907. Calcutta, 1907–1908.

ITALY—MINISTERO DI AGRICOLTURA, INDUSTRIA E COMMERCIO. Revista del Servizio Minerario, 1906, 1907. (2 copies.) Roma, 1907, 1908.

——— MINISTERO DELLE FINANZE. Movimento Commerciale del Regno d'Italia. 1904, Vol. 3; 1907, Vols. 1-2. Roma, 1905, 1908.

——— Movimento Commerciale e della Navigazione di Massaua, 1907. Roma, 1909.

——— Statistica del Commercio Speciale di Importazione e di Esportazione, 1907–1909. Roma, 1907–1909.

JAPAN—BUREAU DE LA STATISTIQUE. Général Résumé Statistique de l'Empire du Japan. 22 Année. Tokyo, 1908.

K. K. ARBEITSSTATISTISCHEN AMTS IM HANDELSMINISTERIUM ARBEITSEINSTELLUNGEN UND AUSSPERRUNGEN IM ÖSTERREICH, 1907. Wien, 1909.

KAISERLICH STATITSISCHEN AMT. Monatliche Nachweise über den Auswärtigen Handel Deutschlands, Dec., 1907. Berlin, 1908.

——— Statistisches Jahrbuch, 1907, 1908. Berlin, 1907, 1908.

——— Vierteljahrshefte zur Statistik des Deutschen Reichs. Year 17, Pt. 2; Year 18, Pt. 1. Berlin, 1908, 1909.

MAGYAR STATISZTIKAI EVKÖNYO. Vols. 14, 15. Budapest, 1908, 1909.

MEXICO—ESTADO Y DEL DESPACHO DE HACIENDA Y CREDITO PUBLICO SECRETARIA. Boletin de Estadistica Fiscal Nos. 300, 327, 334–336. Mexico, 1908, 1909.

MYSORE (INDIA)—GEOLOGICAL DEPARTMENT. Report of the Chief Inspector of Mines, 1904–1905. Madras, 1909.

NATAL—MINES COMMISSIONER. Report on the Mining Industry of Natal, 1906. Pietermaritzburg, 1907.

NEW SOUTH WALES—MINES DEPARTMENT. Annual Report, 1905, 1906, 1907 (3 copies). Sydney, 1906–1908.

NEW ZEALAND—MINES DEPARTMENT. Papers and Reports Relating to Minerals and Mining, 1907, 1908. Wellington, 1907–1908.

——— REGISTRAR GENERAL'S OFFICE. Official Year Book, 1907. Wellington, 1907.

——— Statistics of New Zealand, 1906, Vol. 2; 1907, Vols. 1–2. Wellington, 1907–1908.

NORWAY—STATISTISKE CENTRALBUREAU. Norges Bergvaerksdrift, 1904 og 1905, 1907. Kristiania, 1907, 1908.

——— Statistisk Aarbok, 1907, 1908. Kristiania, 1907–1908.

——— Norges Handel, 1906, 1907.

OBERSCHLESISCHEN BERG UND HÜTTENMÄNNISCHEN VEREIN. Statistik der Oberschlesischen berg und Hüttenwerke für 1908. Kattowitz, 1909.

——— Zeitschrift. Year 48, Feb. and Mar., 1909. Kattowitz, 1909.

PERU—MINISTERIO DE FOMENTO. Boletin del Cuerpo de Ingenieros de Minas. Nos. 54, 63–66, 67 (2 copies). Lima, 1907, 1908.

PRODUCTION DER BERGWERKE, SALINEN UND HÜTTEN DES PREUSSISCHEN STAATES, 1901–1906. Berlin, 1902–1907.

QUEENSLAND—MINES SECRETARY. Annual Report, 1906, 1907 (2 copies). Brisbane, 1907, 1908.

SOCIEDAD NACIONAL DE MINERIA. Estadistica Minera de Chile en 1904–1905, Vol. 2; 1906–1907, Vol. 3. Santiago de Chile, 1907, 1909.

SOUTH AUSTRALIA—MINES SECRETARY. Review of Mining Operations in the State of South Australia during the Half-Year ended June 30, 1908. Adelaide, 1908.

STEINKOHLENBERGBAU DES PREUSSISCHEN STAATES IN DER UMGEBUNG VON SAARBRÜCKEN. Vol. 1, Pts. 1–2; Vol. 2, pts. 4–5. Berlin, 1904.

SVERIGES HANDEL, ÅR 1907 (2 copies). Stockholm, 1908.

SWEDEN—SVERIGES OFFICIELLA STATISTIK:

(*c*) Bergshandteringen. Kommerskollegii underdåniga Berättelse, 1906, 1907. Stockholm, 1907–1908.

(*f*) Handel, 1906, 1907. Stockholm, 1907, 1909.

TRANSVAAL—MINES DEPARTMENT. Annual Report of the Government Mining Engineer, 1907. Pretoria, 1907.

VICTORIA—MINES SECRETARY. Annual Report, 1906, 1907. Melbourue, 1906–1907.

WESTERN AUSTRALIA—GEOLOGICAL SURVEY. Bulletin No. 21. Perth, 1906.

——— MINES DEPARTMENT. Report, 1906–1907 (2 copies). Perth, 1907, 1908.

——— STATISTICIAN'S OFFICE. Statistical Register, 1904. Perth, 1907.

ZEITSCHRIFT FÜR DAS HÜTTEN UND SALINEN WESEN IM PREUSSISCHEN STAATE. Vol. 56, 1 Statistische Lieferung. Berlin, 1908.

TRADE CATALOGUES.

AMERICAN COAL WASHER CO., Alton, Ill. Bulletin No. 10, July 1909. Photographs and description of the Superior coal-washing plant at Gillespie, Ill., equipped with coal-washing machinery manufactured by the above company.

E. C. FLADER, Johstadt, Saxony. Life-saving apparatus for use in mines, as mine fire-apparatus, ambulances, hydrants, fire-hose, etc.

GENERAL ELECTRIC CO., Schenectady, N. Y.

Bulletin No. 4394 C, May, 1909. Form "P" Belt-Driven Alternators, with description and cuts of parts and dimensions.

Bulletin No. 4666, May, 1909. Improved type "H" Transformers, showing construction, advantages, capacity, and tables of dimensions.

Bulletin No. 4670, May, 1909. Gaskets and Bell-Mouths for conduit-wiring, marine, railway-car equipment, and underground work.

Bulletin No. 4674, June, 1909. Polyphase Induction Motors, giving construction, improvements, advantages, and styles.

Type "DLC" Commutating Pole Motor, showing detail construction, styles, and specifications for same.

May, 1909, Index of Bulletins published by the General Electric Co.

Remarkable operating-record of Aluminum Lightning-Arresters reducing number of shut-downs by 88 per cent.

GENERAL ELECTRIC CO., Schenectady, N. Y.

The new Edison 2 candle-power, 10-Watt Sign-Lamp.

A revolution in lighting due to a new lamp, the Tungsten Lamp.

White core, 30 per cent. Para Wires and Cables, N. E. Code thickness of insulation.

National Electric Code standard Wires and Cables and Lamp-Cords.

Tricoat Wires and Cables; N. E. Code thickness of insulation.

Automatic Generator Voltage-Regulation, showing charts of results accomplished.

H. A. HUELSENBERG, Freiberg, Saxony. Pamphlet showing cross-section of part of High-Pressure Centrifugal Pump.

MANISTEE IRON WORKS CO., Manistee, Mich. Manistee Steam Valveless Duplex Pump, showing different sizes, and giving dimensions and data.

NEW YORK ENGINEERING CO., New York, N. Y. Pamphlet on the Empire Hand Prospecting Drill, showing method of drilling, drilling boulders, pulling casing, cost of operation, gold-dredges, and steel-dredges.

ORE CONCENTRATION CO., London, England. Various applications of the Elmore Vacuum Process, and articles from mines in England, Sweden, Africa, and Norway, where the process has been successfully used.

PACIFIC FOUNDRY CO., San Francisco, Cal. Pamphlet showing cuts, and giving description and operation of the new Matte Tapping Car.

ROBINS NEW CONVEYOR CO., 72 Front Street, New York, N. Y. New Robins Genuine Balata Belting for power-transmission and conveying, as well as various grades of Belting for other conveying.

STURTEVANT MILL CO., Boston, Mass. Rock- and Ore-Breakers for rough work; Rock- and Ore-Smashers for intermediate crushing; Plate-Steel Roll-Jaw Crushers for fine crushing; Quadruplex Plate-Steel Crushers.

SWEETLAND FILTER PRESS CO., Los Angeles, Cal. A new pressure filtration machine. The filter is less complicated and bulky than others, and has a greater capacity for its size than any filters made heretofore by this company.

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members or associates who accepted election during the month of August, 1909:

Members.

Edmund S. Dickinson, ... Florence, Wis.
Frank W. De Wolf, ... Urbana, Ill.
Paul Grammel, ... Sumatra, Dutch East Indies.
George G. Greene, ... Washington, D. C.
Edward C. Hugon, ... Hautes Pyrénées, France.
Robert McCart, Jr., ... Indé, Durango, Mex.
Bartolomé Novoa, ... Lima, Peru, S. America.
William B. Patrick, ... Rico, Colo.
Thomas V. Reeves, ... Alameda, Cal.
Harold Schroder, ... Port Kembla, N. S. W., Aust.
W. A. Underhill, ... New Market, Tenn.
Walter S. Weeks, ... Cambridge, Mass.

Associate.

Harold O. Hammond, ... New York, N. Y.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed for election as members of the Institute during the month of August, 1909. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

Fred Howe Bostwick, ... Denver, Colo.
Samuel W. Cohen, ... Cobalt, Ontario, Can.

J. H. Ivey, ... Quechisla, Bolivia, S. Amer.
Charles N. Lindley, ... New York, N. Y.
Alexander Longwell, ... Toronto, Ontario, Can.
Atholl Francis McEwen, ... Cobalt, Ontario, Can.
Joseph B. S. McIntosh . . . . Tooele, Utah.
Albert Daniel Oberly, ... Scottdale, Pa.
John Orr, ... Johannesburg, S. Africa.
John E. Penberthy, ... Globe, Ariz.
John G. Perry, ... Denver, Colo.
Charles H. Schlacks, ... Denver, Colo.
William Cowper Stratton, ... Scottdale, Pa.
Lewis Arthur Westcott, ... Miriam Vale, Queensland, Aust.
Rush J. White, ... Wallace, Idaho.

Associate.

James Finch Callbreath, Jr., ... Denver, Colo.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of August, 1909. This list, together with the lists given in the *Bulletin*, Nos. 26 to 32, for February to August, therefore, supplements the annual list of members corrected to Jan. 1, 1909, and brings it up to the date of Sept. 1, 1909. The names of Members who have accepted election during the month (new members), are printed in *italics*.

ADAMS, ARTHUR K., U. S. Mineral Inspector, General Land Office, Santa Fé, N. M.
ALABASTER, RUPERT C. .... Oonah Mines, Ltd., Zeehan, Tasmania.
ALMY, WILLIAM F. .... P. O. Box 204, Nashua, N. H.
ATKIN, AUSTIN J. R. .... Box 134, East Rand, Transvaal, S. Africa.
BAINBRIDGE, WILLIAM H. .... P. O. Box 362, Santa Monica, Cal.
BANKS, NOBLE C., Care Gear Grinding Machinery Co., Boydell Bldg., Detroit, Mich.
BATCHELLER, JAMES H., Care Wm. DeY. Field .... Mattapoisett, Mass.
BAYLES, FREDERICK P. .... Cokedale, Colo.
BOSQUI, FRANCIS L., Care H. Eckstein & Co., Johannesburg, Transvaal, So. Africa.
BRALY, NORMAN B. .... 845 W. Silver St., Butte, Mont.
BRAYTON, COREY C. .... 309 Clumie Bldg., San Francisco, Cal.
BRODIE, WALTER M., Batopilas Min. Co. .... 45 Broadway, New York, N. Y.
BROWN, HARVEY S. .... Yerington, Nev.
BUCKLEY, ERNEST R. .... Rolla, Mo.
BURGER, CLARENCE C. .... 71 Broadway, New York, N. Y.
BYRNES, OWEN .... Flesher, Lewis & Clarke Co., Mont.
CAZIN, FRANZ, Mech. Engr .... 405 Jackson Bldg., Denver, Colo.
CLELAND, E. DAVENPORT, Inspector of Mines, Care Dept. of Mines, Perth, West Australia.
CLERC, F. L. .... 516 Mapleton Ave., Boulder, Colo.

COLLINS, W. J................................2106 Independence Ave., Kansas City, Mo.
DARGIN, PERCY W...General Delivery, Denver, Colo.
DAY, ARTHUR M...P. O. Box 382, Thermopolis, Wyo.
DEVEREUX, WALTER B., Cons. Min. Engr...........64 Wall St., New York, N. Y.
DE VORE, E. H., International Portland Cement Co., 1327 Commerce Bldg., Kansas City, Mo.
**De Wolf, Frank W.*, Geol., State Geological Survey...............Urbana, Ill. '09.
**Dickinson, Edmund S.*, Min. Engr.....................................Florence, Wis. '09.
DOWE, JOHN H.........................Care Rose Mount, Wallington, Surrey, England.
DURANT, HENRY T., British Metals Extraction Co., Ltd., Llansamlet, Glamorganshire, So. Wales, G. B.
DYSON, THOMAS I., Met., Care Aust. Inst. of Min. Engrs., 57 Swanston St., Melbourne, Vic., Aust.
EVELAND, ARTHUR J., Care General Development Co., 42 Broadway, New York, N. Y.
FAWCET, JAMES H., Min. Engr...........31 Queen St., Melbourne, Vic., Australia.
FITCH, MAX B...1811 Wilton Place, Los Angeles, Cal.
FITZGERALD, J. MORTON...............................149 Broadway, New York, N. Y.
FRIZZELL, PORTER T., Min. and Met., Compania Minera de Rio de Plata, Guazapares, Chih., Mexico.
FROSSARD, JOHN D...Orthez, Basses-Pyrénées, France.
GARTHWAITE, E. H., Butters Copala Mines...333 Kearny St., San Francisco, Cal.
GEISMER, HENRY S.............Care Southern Iron & Steel Co., Chattanooga, Tenn.
GIFFORD, ALVAH W..Instructed to hold all mail.
GILMAN, CHARLES E........................Humboldt Bank Bldg., San Francisco, Cal.
GRABILL, CLARENCE A.................................909 Montana St., El Paso, Texas.
**Grammel, Paul*, Mgr., West Sumatra Mijnen Synd., Fort-de-Kock, Sumatra, Dutch East Indies. '09.
GRAY, JAMES H., Care U. S. Steel Corp., Room 1605, 71 Broadway, New York, N. Y.
**Green, George G.*, Min. Engr......133A Vermont Ave., Washington, D. C. '09.
GREGSON, WILLIAM H..Instructed to hold all mail.
†*Hammond, Harold O.*, Met. and Min. Engr., 203 W. 109th St., New York, N. Y. '09.
HARDENBERGH, WILLIAM P.............................55 Wall St., New York, N. Y.
HECK, ELMER C..............................Hotel Arcadia, Hermosillo, Son., Mexico.
HOLLIS, GEOFFREY C....................................Apartado 57, Pachuca, Mexico.
HOLLISTER, JOHN J..815 15th St., Sacramento, Cal.
HORSWILL, FREDERICK A.................................1070 16th St., Oakland, Cal.
HOWE, FRANK P.....................................260 Drexel Bldg., Philadelphia, Pa.
HUDSON, ALBERT W...............Met., Great Cobar, Ltd., Cobar, N. S. W., Aust.
**Hugon, Edward C.*, Min. Engr., Mines de Pierrefitte, Hautes Pyrénées, France. '09.
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WOODBURY, FRANK E.............................Colby-Abbot Bldg., Milwaukee, Wis.
WOODMAN, J. EDMUND......................New York University, New York, N. Y.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address on Records, from which Mail has been Returned.
Adams, Randolph,	Copperhill, Tenn.
Alexander, George E.,	Sparta, Ore.
Allen, Frederick E.,	Bloomsburg, Pa.
Andrew, Thomas,	Pretoria, So. Africa.
Arozarena, R. M. de,	Mexico City, Mexico.
Bartoccini, Astolfo,	214 E. 90th St., New York, N. Y.
Bassett, Thomas B.,	Cumpas, Sonora, Mexico.
Batchelder, Joseph F.,	54 1st St., Portland, Ore.
Bellam, Henry L.,	Reno, Nev.
Bouchelle, James F.,	22 Duncan Ave., Jersey City, N. J.
Brook, Henry E. C.,	Cadia, N. S. W., Australia.
Brown, Frank H.,	Coppermount, Alaska.
Campa, Jose,	Mexico City, Mexico.
Cragoe, A. Spencer,	Vencedora, Mexico.
Derby, Harry S.,	134 Monroe St., Chicago, Ill.
Dickson, George H.,	Lethbridge, Alberta, Canada.
Dougherty, Clarence E.,	41 Wall St., New York, N. Y.
Ekberg, Benjamin P.,	Johannesburg, Transvaal, So. Africa.
Field, Wilfrid B.,	Mexico City, Mexico.
Fitzsimmons, F. J.,	Cananea, Mexico.
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Fuller, Frederick D.,	Sumpter, Ore.
Gage, Edward C.,	San Dimas, Dur., Mexioc.
Gee, Emerson,	Reno, Nev.
Hawkins, Tancred,	Ballydehob, Ireland.
Hunt, Thatcher R.,	Iron Mt., via Keswick, Cal.
Jackson, Byron N.,	Milton, Cal.
Jessop, Herbert J.,	Guanacevi, Mexico.
Jewett, Eliot C.,	2918 Morgan St., St. Louis, Mo.
Judd, Henry A.,	Mertondale, W. Australia.
King, Rufus H.,	Union Club, New York, N. Y.
Kow, Tong Sing,	Shanghai, China.
Mildon, Reginald B.,	Nacozari, Son., Mexico.
Moulton, Herbert G.,	Cobalt, Ont., Can.

Muir, Thomas K.,	Portland, Ore.
Nawatny, William F.,	Harrisburg, Ill.
O'Byrne, Joseph F.,	Midas, Nev.
Philbrick, Arthur,	Manhattan, Nev.
Piper, John W. H.,	Buenos Ayres, Argentine Rep., S. A.
Potter, J. A.,	41 W. 124th St., New York, N. Y.
Rigney, Thomas P.,	Reno, Nev.
Rodda, Richard W.,	Seattle, Wash.
Sandifer, Harmer C.,	El Oro, Mexico.
Schlemm, William H.,	Durango, Mexico.
Scott, Winfield G.,	Long Beach, Cal.
Skelding, Joseph F.,	Embreeville, Tenn.
Thomas, Richard A.,	43 Wall St., New York, N. Y.
Vaux, Charles A.,	P. O. Box 80, East Rand, So. Africa.
Vidler, Louis W.,	Lookout Mountain, Colo.
Warren, Henry L. J.,	Salt Lake City, Utah.
Wiswell, Herbert J.,	Carterville, Mo.
Wolfe, Burton L.,	Ely, Nev.
Young, William,	Kenora, Ont., Canada.

BIOGRAPHICAL NOTICES.

Charles Caspar Mattes was born Aug. 5, 1858, at Scranton, Pa., of which place his father, Charles F. Mattes, was one of the pioneer inhabitants. He was educated in the public schools of Scranton, and at the age of nineteen went to Virginia, where he served for a year as an assistant engineer under his elder brother, William Mattes. Returning to Scranton, he entered as civil and mining engineer the employ of the Lackawanna Iron & Coal Co., with which he remained for more than thirty years. During the first half of this period a large part of his work consisted in subterranean surveys, which were often exceedingly dangerous, involving the running of lines through abandoned workings, where even the experienced foremen and fire-bosses hesitated to venture. As mining engineer of the company's Tilly Foster mine he devised an ingenious apparatus for measuring open pit-work. The problem was to determine as quickly as possible, at the end of each month, the yardage excavated by the contractor; and he solved it by means of a trolley, traveling on a light, suspended, movable cable, while the surveying engineer, with his instrument, remained on the bank of the excavation and read off the varying depths and distances. (See the paper by F. H. Macdowell on The Reopening of the Tilly Foster Iron Mine, *Trans.*, xvii., 758, in which the author says that this device will prove invaluable as depth increases, and is

worthy to be the subject of a special paper.) He became a member of the Institute in 1887.

Upon the death of his father, in 1895, Mr. Mattes succeeded to the general management of the company's property, and acquired in that position the highest reputation as an authority concerning the extensive boundary-lines and innumerable subdivisions of that immense estate, as well as an inspector and judge of the construction of buildings—in which latter capacity he was chosen by his fellow-citizens to serve upon such important edifices as the Albright Library, the First Presbyterian Church, and the great armory of the Thirteenth Regiment of the National Guard of Pennsylvania. He was often summoned as an expert witness on land-divisions and real-estate values.

In 1878, he enlisted in the Scranton City Guards, a battalion which afterwards became the Thirteenth Regiment of the National Guard. In this regiment he served for twenty-one years, passing through all grades from private to lieutenant-colonel. Militia service in Pennsylvania is not a mere matter of sham fights and parades. During the Homestead riots of 1892 Mr. Mattes performed the double and dangerous duty of regimental quartermaster and commissary of subsistence. In the Hazleton campaign of 1897, following the riot and bloodshed at Lattimer, he was in command of his regiment. In the Spanish war of 1898, the regiment having offered its services to the government, he was commissioned as Lieutenant-Colonel of the Thirteenth Infantry, Pennsylvania Volunteers, and acted as the regimental commander during the trying experiences at Fort Alger, Va. After the war he was enrolled as Colonel on the list of the retired officers of the Pennsylvania National Guard, being rated also as a veteran sharp-shooter.

Colonel Mattes held high Masonic rank, and was a member of several military and engineering societies. He died June 8, 1909, at his home in Scranton, after an illness of little more than three weeks, originating in a cold which developed into pneumonia. His wife, a son and three daughters survive him, and the people of Scranton mourn his loss as that of a public-spirited leader, an upright Christian, and a loyal friend.

James Stirling was born Jan. 9, 1852, at Geelong, Victoria, Australia. His parents had come to Australia from Scotland

with their respective families at the time of the rush to the "diggings." He was the first white baby born at Geelong, and, when about three months old, was stolen by the "blacks" while asleep in his cot on the veranda. A few hours later his father and friends succeeded in tracking the tribe, and found the infant wrapped in opossum-skin in one of the women's "*mia-mias*," or rude tents. He was educated at the Geelong College until the age of fourteen, when his father died, and since he was at that time rather delicate in health, his mother thought life on his uncle's cattle-station in Gippsland would be beneficial to him. He there renewed his intercourse with the natives, generally in a friendly fashion, but sometimes otherwise—as was testified by the many spear-scars he bore. On one occasion he traveled down the Snowy river for a fortnight, lying in the bottom of a canoe with a broken shoulder-blade and other injuries, and only a "black fellow" to manage the boat, provide the fish, and cook it. He had an intimate knowledge of the habits and languages of the Australian native tribes, now rapidly becoming extinct. He studied for a time at the School of Mines, Ballarat. When his college-days were over, his mother articled him to a well-known firm of architects. But his health suffered under the close office work, and when a position as surveyor was offered him by the government, the firm generously canceled his articles. In a very short time his ability, marked integrity of purpose, and high moral character drew to him the attention of the Minister of Lands, and at the age of twenty-three he was appointed District Surveyor and Lands Officer for Omeo, a large mountainous mining and agricultural district. While attending assiduously to the duties of his office (which was by no means a sinecure, often requiring him to spend days in the saddle, sometimes riding over tracks where there was scarcely a footing for man or beast), he lost no opportunity of studying both by books and observation botany, geology, petrography, and kindred subjects, and his writings on his original researches in these fields attracted to him the attention of the leading scientists of that period in Australia, many of whom became his correspondents.

When a vacancy occurred on the Geological Staff of the Mines Department it was offered to him; and the real work of his life began. Coal had already been discovered by Paterson

and others in the bay and along some parts of the coast; but Mr. Stirling, from observation and deduction, felt convinced that the most valuable deposits would be found in the dense forests of Gippsland, and persuaded the Hon. R. Outrim, then Minister of Mines, to equip him with a party to penetrate the hitherto untrodden recesses. At first only one assistant was alloted to him; and, axe in hand, he and his companion literally hewed their way through the thick semi-tropical undergrowth until Mr. Stirling was able to demonstrate what had till then been only scientific deduction, and the government allowed him a sufficient number of men to carry out a complete geological survey, with the result that the well-known productive coal-fields of Korrumburra, Jumbunua, Morwell, Leongatha, etc., were opened up. His work for the development of all the mineral resources of his country was indefatigable, and was always a labor of love. The gold- and coal-developments especially, of Victoria, owe more than can be estimated to his indomitable energy and masterly knowledge.

In 1885 he was appointed Assistant Government Geologist; and so faithfully and ably did he fulfill his duties that his chief, Mr. R. Murray, was wont to call him his colleague, not his assistant. When Mr. Murray retired Mr. Stirling succeeded him as Government Geologist and Chief Inspector of Mines in Victoria. In 1900 he was sent by the government to open an office in London to supply information on the resources of Victoria, and to investigate the latest methods of mining, machinery, etc., in Europe. For three years he worked with his usual energy, never sparing himself, lecturing in all parts, and traveling in Europe only for the purpose of acquiring knowledge relating to mines and mining. The heavy fogs of London affected his throat and lungs, and he asked to be relieved from the post.

On his return to Australia in 1901 he retired to a beautiful country home, intending to carry on a private practice; but when, four years ago, a forest-fire destroyed his property, leaving nothing but black desolation behind it, he decided to come to California, a country he had always wished to see. There he took up his profession as geologist and petrologist with all his natural enthusiasm, and made many valuable reports on mineral properties, besides winning many new friends, whom he loved and appreciated. He died June 26, 1909, in Riverside, California.

Professionally he was known in California, Arizona, Nevada, and Utah, where he made extensive examinations for various companies. His writings are mainly embodied in the form of reports and lectures; the former are so comprehensive and at the same time so clearly expressed that they are even to the ordinary reader interesting literature; and the latter, comprising lectures on crystallography, petrography, palæontology, caves, etc., are the epitomized reflections and thoughts of one who never dealt with any subject superficially, and who had in a peculiar degree the faculty of keeping in touch with and culling the best from the great thinkers of the day; indeed, so alert and receptive was his mind that he seemed to walk abreast with each new scientific discovery.

Mr. Stirling was a Fellow and an ex-President of the Australasian Institute of Mining Engineers, and contributed a number of papers to its *Transactions;* he was an ex-President of the Geological Society of Australasia; a member of the Council of the Australasian Association for the Advancement of Science, and the Victorian Chamber of Mines; a member of the Royal Society of Victoria, the Geographical Society of Australasia, the Victorian Institute of Surveyors, the British Association for the Advancement of Science, and the Royal Institution, London; a Fellow of the Royal Geographical Society, and an associate of the Institution of Civil Engineers, London. He became a member of the American Institute of Mining Engineers in 1906.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Concentration of Silver-Lead Ores at the Works of Block 10 Co., Broken Hill, N. S. W., Australia.*

BY V. F. STANLEY LOW, M.M.E., BROKEN HILL, N. S. W., AUSTRALIA.

(New Haven Meeting, February, 1909.)

INTRODUCTION.

THERE is not the slightest doubt that the present recoveries of valuable minerals by the Broken Hill mills could be improved, and that further machinery would be installed for the purpose if it could be assured that a high price of metals would be maintained; but concentration, like all other commercial propositions, is a question of profit and loss, and no sensible metallurgist would carry out his work in such a manner as to obtain high recoveries at a financial loss to his company. There is a point in all concentration at which the added money obtained from the higher recovery of minerals is balanced by the extra cost of obtaining the last part of the recovery; a wise metallurgist will be content with his work before his working-expenditure reaches this point.

To a casual observer, the work being done by the various Broken Hill mills is identically the same. A closer investigation will show that this is far from being the case, for it is a remarkable fact that, although the larger mines extend over a length of approximately only 2 miles, such a difference in the physical and chemical properties of the ores exists that, from the first reduction downwards, special treatment is necessary in each case, and hardly any two mills are identical in operation.

The details of this paper refer directly to the work of one particular mill in Broken Hill, for the reason that the ore from each mine presents special difficulties and requires its own special treatment. I have selected the mill referred to because it has been constructed and modified under my supervision.

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* This paper in a more extensive form has also been published, by permission of the Council of the Institute, in the *Australian Mining Standard*, Mar. 31, Apr. 7, 14, and 21, 1909.

Character of Ore.

The ore from the Block 10 mine is drawn from depths varying from 315 to 1,465 ft., and is ever varying in its nature. Soft, friable ore is found resting immediately upon very hard high-grade material, which, in its turn, abuts on low-grade ore mixed with much rhodonite, the whole carrying large quantities of zinc. In the lower levels the ore is fine-grained and carries more than 20 per cent. of zinc, necessitating very fine comminution before the lead is set free. The fine reduction necessarily makes much slime, and it is remunerative to treat only a portion of this slime by ordinary hydraulic methods. Much of the silver contained in the ore is allied with the zinc, and must be stored away with the mill by-products until such time as it is found profitable to make a zinc-concentrate. Although only a comparatively small proportion of silver is recovered in the lead-concentrates, the remainder of it cannot be regarded as being lost, but rather as being held in suspense, for recoveries as high as 90 per cent. of the silver-content of the stored zinkiferous by-products have been made by processes recently introduced for the production of zinc-concentrates.

The average analysis of the crude ore, at present being sent to the concentration-mill for treatment, is:

		Per Cent.
Silica,	23.8	36.0
Rhodonite,	12.2	
Lead,		14.4
Zinc,		19.4
Iron,		4.6
Manganese (MnO),		3.2
Alumina,		2.8
Carbon dioxide,		3.3
Calcium oxide,		3.5
Sulphur,		12.1
Total,		99.3

Outline of the System Employed.

The following is a brief outline of the sequence of operations, which is further illustrated by Figs. 1 and 2. The details of the several different machines will be referred to later. The ore, broken on contract in the underground workings, is sent to the brace in trucks holding from 15 to 22 cwt. Immediately

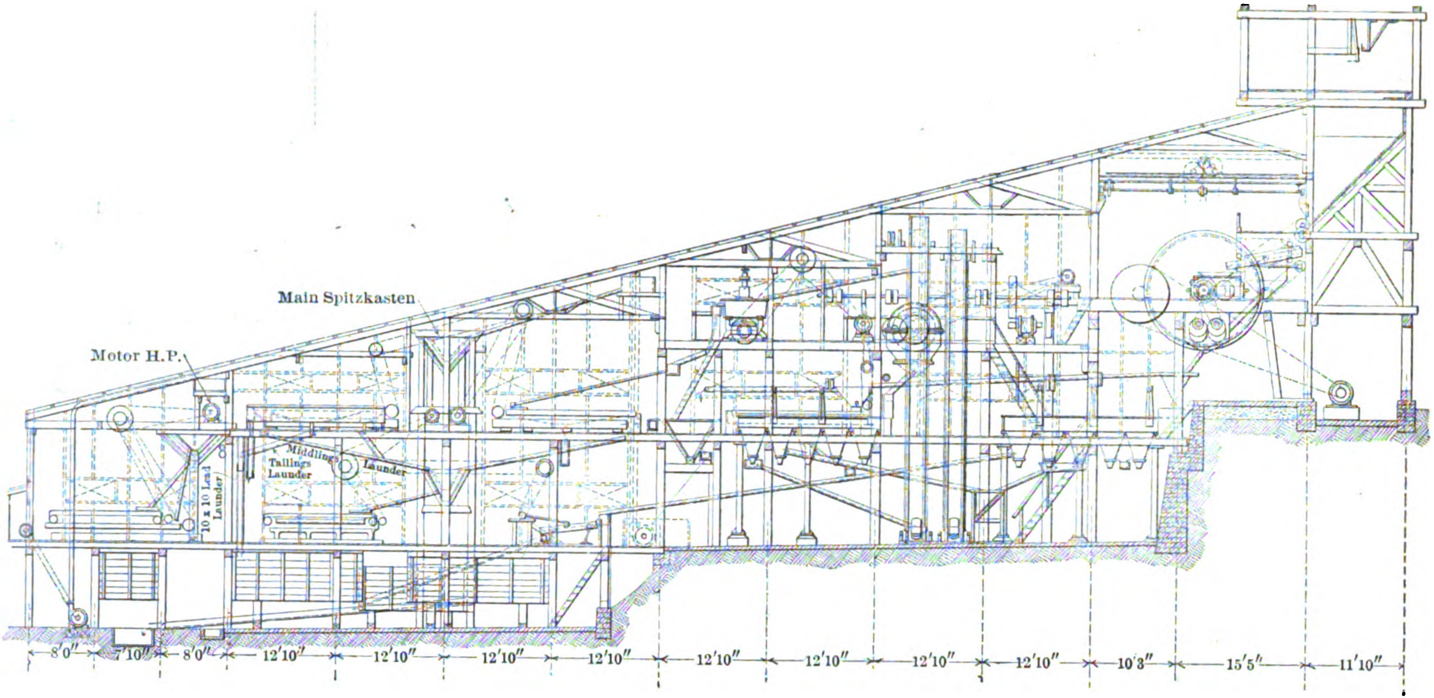

FIG. 1.—SIDE-ELEVATION OF CONCENTRATING-MILL.

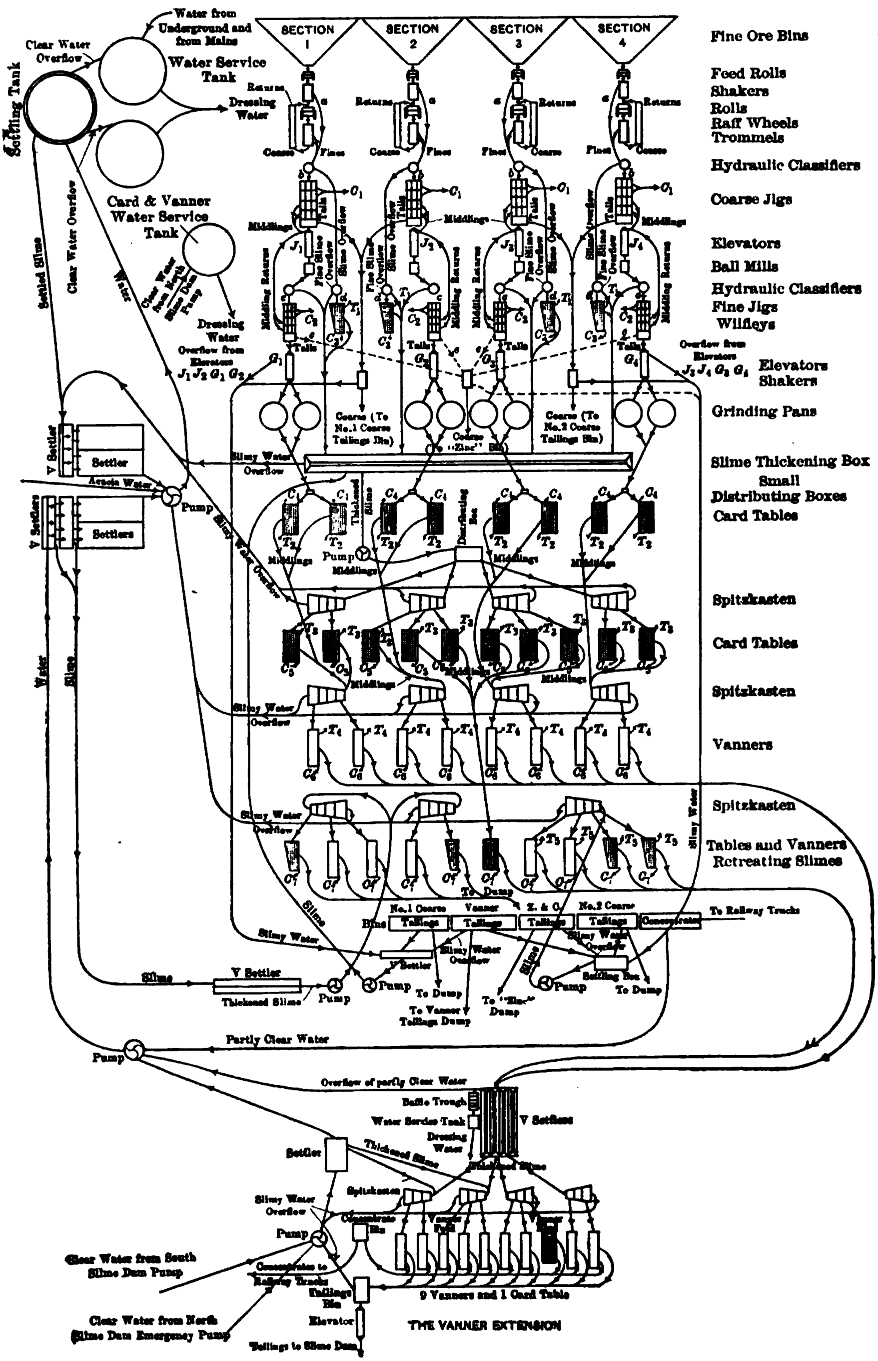

FIG. 2.—FLOW-SHEET OF LEAD-CONCENTRATING MILL.

on leaving the cage the trucks are run on to a weigh-bridge, and the net weight of the ore contained is credited to the party of contractors by whom the ore has been mined. For the purpose of identification, a metal pocket is attached to the side of each truck, and in this pocket a ticket bearing the number of the contracting party is placed before the truck leaves the underground department. While the truck is on the bridge, the ticket is handed to the weigh-man, who makes a record of the number of the party and the weight of the ore. Ore-breaking contracts are set at so much per ton for fortnightly periods, and these contracts stipulate that no material sent to the surface shall measure more than 10 by 8 inches.

The weighing having been completed, the ore is tipped over grizzlies for the purpose of separating the fines from the larger material, and the whole passes to bins of holding capacity equal to 14 hr. mill-supply. Placed immediately below the bins containing the oversize from the grizzlies are large breakers, which reduce the ore to 1.25-in. ring-gauge. Below the breakers are small supplementary bins for holding the crushed material. From these supplementary bins, and from the bins holding the through size from the grizzlies, the ore is conveyed by an aërial tram to the mill, where it is deposited in bins which have a capacity equal to 17 hr. mill-supply. From the mill-bins, the ore is delivered by means of feed-rollers to shaking-screens, which separate the finest material from the larger, and the oversize from the screens passes on to crushing-rolls and thence to the trommels. The oversize from the trommels is returned to the rolls by means of raff-wheels, while the through size from the trommels meets the through size from the shaking-screens, and together they are conveyed by launder to the hydraulic classifiers at the heads of the coarse jigs. While the ore is being delivered by the feed-rollers, jets of water are brought into contact with it, and, thence onwards, water is used for all separation-purposes.

The fine material and the slimes are eliminated from the jigging-material in the classifiers and are carried to further classifiers at the heads of the Wilfley tables. The jigging-material is delivered from the classifiers to the coarse jigs, which produce concentrates, middlings, and tailings. The concentrates go direct to the concentrates-bins; the tailings,

after passing over shaking-screens, which remove the finest material for further treatment, go to the tailings-bins; while the middlings are raised by means of bucket-elevators to ball-mills for further size-reduction. The material delivered from the ball-mills is conveyed by launders to the hydraulic classifiers at the heads of the fine jigs. The overflowing fines from these classifiers and from the classifiers attached to the Wilfley tables are carried to settlers for thickening and further treatment. The fine jigs produce concentrates, middlings, and tailings. The concentrates produced, together with those from the Wilfley tables, are conveyed by launder to the concentrates-bins; the middlings are returned to the ball-mills for further crushing, and eventually find their way to the same jigs, and the tailings are sent to the grinding-pans for comminution. The tailings from the Wilfley tables are collected in the tailings-bins, while the middlings are carried away for further treatment.

The material leaving the grinding-pans is fed, without further classification, on to Card tables, which produce concentrates, tailings, and middlings. The first two find their way to their respective bins, and the last is collected for further treatment. The overflow from the classifiers attached to the fine jigs and Wilfley tables goes to a large settler situated at about the center of the mill, and the overflow water from the settler gets a further clarifying in outside settlers before being pumped back to the mill-service tanks for further use. The thickened material from the interior settler is delivered by launder to a centrifugal pump, which elevates it to *spitzkasten*, whence, after classification, it is fed to Card tables. The middlings from these tables, together with those from the Wilfley tables and Card tables previously mentioned, are in turn classified in *spitzkasten* and treated on Warren belt-vanners.

The bottom or trucking-floor of the mill is given up to pumps, settlers, and bins for concentrates and tailings. On this floor the concentrates are trucked from their settling-bins to the dispatch-bins situated at the railway-siding; the tailings are removed from their various settling-bins to their respective dumps. The overflow from the settling-bins, together with all water reaching this floor, goes to settlers for further clarification; the thickened material from the settlers, which also includes the fine material from the jig-tailings shaking-

screens, is sent to a separate vanning-plant for final treatment, and the clarified water is returned to the mill-service tanks. The rejected material from the final vanner-plant is thickened, elevated, and dispatched by launder to the slime-dumps. The surplus water is returned to the mill-service tanks. The water overflowing from the main settler and all *spitzkasten* in the mill proper passes for further clarification to outside settlers, and is thence pumped to the mill-service tanks. Some of the slimes deposited by these outside settlers have successively passed over every class of machine and classifier in the mill, but they are given yet one more chance of treatment on slime-tables of various kinds before being finally rejected and relegated to the slime-dump.

The slime leaving the various parts of the mill for its final deposition in dumps necessarily carries with it a considerable quantity of water. By raising a bank round the edge of the slime-dumps the water that is not lost by seepage and evaporation becomes clarified by the deposition of the slime, and is pumped back to the mill-service tanks. It is a notable fact that in not one mill on the Broken Hill field is there installed either a belt or a table for hand-picking the ore prior to mechanical concentration. The absence of hand-picking or sorting in Broken Hill is accounted for by the usually homogeneous nature of the material to be treated, but it is questionable whether the practice might not be advantageously adopted in one or two of the plants.

Details of Operation.

Grizzly.—The grizzly used for separating the coarse from the fine material is composed of manganese-steel bars, 12 ft. long, 4 in. deep, $\frac{7}{8}$ in. wide at the top and $\frac{7}{16}$ in. wide at the bottom. These bars are held apart by circular washers, so that they are $\frac{3}{4}$ in. distant on the upper side and $1\frac{1}{8}$ in. distant on the under side. Four bolts, 1 in. in diameter and 3 ft. 4 in. apart, pass through the bars and washers. The grizzly is 4 ft. in width and has an angle of inclination of 45°. The opening between the bars is made smaller on the upper side in order to prevent clogging.

Primary Reduction.—Primary reduction at the Broken Hill mines is generally performed by gyratory crushers. In most

cases the material from the first crushing goes direct to the roll-bins, but in some cases the ore receives a further reduction by small jaw-crushers before being passed on to rolls. On the Block 10 mine both gyratory and jaw-crushers have been tried, but the former have proved the more suitable for the duty to be performed. There are at least three different classes of gyratory crushers on the field, the Hadfield, the Gates, and the Austin, in all of which the greatest movement is on the smallest lump. Austin crushers are installed at the Block 10 works, and vary from the other makes in having continuous lubrication for the eccentric by means of an automatic oil-pump, in having the eccentric above instead of below the bevel-wheel, and in having an extra bearing for the counter-shaft, whereby the bevel-pinion is supported on both sides instead of having an overhang, as in the Gates type. It is also claimed that, owing to the shape of the head, the crusher can deal with just as large stones in crushing to 1-in. as in crushing to 2.5-in. size.

The principal wear on all gyratory crushers dealing with very hard ore is on the eccentric, which is usually made of cast-iron babbitted on the wearing-surfaces. In both the Austin crushers now installed and in the Gates crushers originally in use at this mine, this trouble was so insistent that eccentrics made of solid brass were tried, and they proved so much more satisfactory that they have since been in constant use.

Another cause of trouble was the stripping of the teeth on the bevel-pinion, but this has been reduced to a minimum by making the pinion also of brass. There is considerable wear on the upper bearing of the spindle situated within the spider. The present practice is to place false bushings inside the bushing proper, made of sheet-steel from $\frac{1}{16}$ to $\frac{1}{8}$ in. in thickness, and bent to the required shape; when worn out these false bushings are easily replaced. It is now usual to make both head and concave of smooth manganese-steel, thus insuring more even and finer crushing, as well as longer life to the head, than was the case with corrugated heads.

Installed also is a 30- by 18-in. Blake crusher, which is now used only when one of the gyratory crushers is under repair. Running for an extended period under similar working-conditions against a gyratory crusher on similar classes of ore, it was found that the jaw-crusher consumed more lubricant, required

more attention for feeding, cost more for repairs, and was generally less satisfactory than the gyratory crusher.

The gyratory crushers are of the No. 5 Austin class, in which the pinion-shaft makes 370 rev. per min., at an average consumption of from 25 to 30 b.h.p. per hour in crushing 25 tons of material per hour to 1¼-in. gauge. The actual horse-power consumed per ton of ore crushed to this size is greater than that required for crushing material to, say, 2½-in. gauge, but this is on account of the greater length of time required for doing the finer work. In crushing finely a smaller motor or engine may be used, for, although it will have to work more hours to break the same weight of material, the amount of horse-power being expended at any particular moment will be considerably less than in crushing to a large size. The jaw-crusher in crushing fairly soft material to 1¼-in. gauge also takes from 25 to 30 h.p., with the crank-shaft running at 205 rev. per minute.

Aërial Tram.—The aërial tram, made by Pohlig, of Cologne, consists of a loading- and an unloading-station and two carrying-ropes stretched and supported at intervals on four steel-lattice standards; the two stations are constructed of Oregon pine. The rope carrying the loaded buckets is of the lock-strand type, 36 mm. in diameter; the rope carrying the empty buckets was originally an ordinary unstranded steel-wire rope of 22 mm. diameter, but has since been replaced by a lock-strand rope. The buckets are hauled by an endless steel Lang-lay rope, 13 mm. in diameter, supported on sheaves. This rope is driven by a 12-h.p. electric motor, and has the usual tension-carriage attached to it; the tension-carriage is kept in place by a hanging weight of 1 ton. The standards vary in height, the highest being 60 ft. from the ground, and the longest span between supports is 850 ft. The carrying-ropes are made fast at the unloading-station, at a height of 40 ft. above the surface, while at the loading-station their ends are coupled to flexible steel-wire ropes passing over sheaves and loaded with tanks containing weights of 18 tons and 9 tons. On entering each station the buckets pass from the carrying-ropes on to steel rails, suspended from the timber-work by cast-iron hangers. Each bucket carries 0.5 ton of broken material, and is pivoted on the frame-work, which hangs from a pair of steel grooved runners of small wheel-base. As the pivots are placed so that

the line joining them passes without the center of gravity of the bucket, when the catch which keeps it in upright position is released the bucket readily tips and discharges its contents. The bucket dumps automatically on coming into contact with a bumper placed in such a position as to throw back the catch as the bucket passes. The gripping and uncoupling are both automatically effected as the bucket passes out of and into the station. The line has a total length of about 2,000 ft., and the buckets travel at the rate of about 5 ft. per second.

Feed-Rollers.—In the bottom front of each mill-bin is an opening fitted with a sliding-door, which may be raised or lowered by means of a rack and pinion. Immediately in front of the door is placed a flanged cast-iron roller, 12 in. in diameter and 15 in. wide between the flanges. This roller makes 4 rev. per min., and receives its motion through belt and gear-wheels from the roll-shafting. The operation of the feed-rollers insures regular feed to the shaking-screens, and the quantity delivered may be varied by raising or lowering the side-door.

Shaking-Screens for Crude Ore.—The crude-ore shaking-screens have the sides and back ends constructed of 10- by 2.25-in. Oregon pine, and are suspended by links or rods so as to give the screens an inclination of 10° from the horizontal. The broken ore, upon leaving the feed-rolls, falls on to false bottoms 6 ft. long, the first 2 ft. of which are of plain steel and the remaining 4 ft. of steel-plate punched with holes $\frac{3}{8}$ in. in diameter. The true bottoms, placed below the false bottoms, are composed of plain sheet-steel, $\frac{3}{16}$ in. thick for the first 2 ft. 6 in. and for the last 2 ft. 4 in.; the middle portion, 5 ft. 6 in. in length, is composed of screening-plates punched with round holes, $\frac{1}{8}$ in. in diameter; the plates are 1 ft. 8 in. wide. Motion is obtained from an eccentric through an iron rod attached to the bottom of each screen, and 200 vibrations, 1.5 in. long, are made every minute. The through size goes to the coarse jigs, the oversize to the rolls.

Rolls.—The rolls are of the Cornish geared type, and one roll is flanged, the other plain, the one fitting in between the flanges of the other. The object of having one roll flanged is to keep the faces of the rolls in proper line. If both rolls be unflanged there is often a tendency for the edge of one roll to overlap the edge of the other, and thus not only lessen the

crushing-surface, but also cause an uneven wear on the roll-shells; in order to keep the ends of the rolls in the same plane elaborate precautions are necessary, since otherwise flanges are formed on the overhanging ends, and these flanges must be turned in the lathe and worked down with an emery wheel, or removed in place by a special arrangement of emery block and weights. Each roll-shell has an external diameter of 30 in.; the face of the flanged roll is 15⅝ in. wide, and of the unflanged, 15 in. wide; the flanges are 1 in. deep and 1 in. thick at the base.

Various metals are used for the manufacture of shells—hard cast-iron, toughened steel, manganese-steel, etc. Better results are obtained when one shell is of manganese-steel and the other of cast-iron or of toughened steel. The present practice at the Block 10 mill is to have both shells of toughened steel.

A set of rolls as above described will, making 15 rev. per min. and absorbing from 22 to 26 h.p., crush, per week of 144 hr., from 900 to 1,000 tons of ore of average hardness, from 1.25-in. ring-gauge, to pass through screens perforated with holes ⅛ in. in diameter. These figures refer to single-stage crushing. The horse-power required, the wear on the shells, and the capacity of the rolls, all vary with the nature of the ore to be operated upon. The capacity of the rolls is also greater when the shells are almost new and have a maximum diameter. A large part of the ore treated at Block 10 mill is hard and contains much rhodonite; the wear on the shells is, therefore, great, and the average life of a set of toughened-steel shells is from 14 to 17 weeks in crushing about 750 tons per week. The average horse-power required varies from 22 to 26, but there are momentary variations which increase this amount very considerably. It is notable that the greatest capacity is obtained in the single-stage crushing here practiced when the shells are just lightly in contact, and that the horse-power consumed is very heavily increased with the size of the material sent to the rolls for reduction.

Single-Stage versus Series-Reduction.—In single-stage reduction the oversize material from the trommels is sent back to the same set of rolls through which it has already passed, where it is mixed and recrushed with fresh material from the shaking-screen. In series-reduction only crude ore is sent to the first

set of rolls; the oversize from the first set of trommels goes to a second set of rolls for recrushing; the oversize from the second set of trommels goes for further crushing to a third set of rolls, and so onwards. There are advantages and disadvantages connected with both systems, and whereas it is quite possible that, with one class of ore the advantages of series-reduction may greatly outweigh the disadvantages, it is more than possible that with another class of ore the reverse may be the case. The advantage claimed for series-reduction is that lessened sliming takes place. It is also possible that better trommeling arrangements may be made with this process of reduction.

Some of the disadvantages are:

(1) Increased number of roll-sets required; (2) increased consumption of power; (3) extra elevators required, thereby further increasing the cost of operation and up-keep; (4) extra bin-accommodation necessary; (5) increased water-consumption; (6) increased cost of attendance; (7) as the number of sets of rolls must be increased by at least 50 per cent., the cost of maintenance and repairs is correspondingly increased; and (8) the work of each set of rolls is dependent on that of its neighbors, and all the jigs do not get an equal amount of feed.

Although the disadvantages appear to be very formidable, it is quite possible that the advantages gained in lessened sliming may in some instances more than outweigh them.

Experimental trials were made with a view to discover whether series-reduction might be introduced advantageously into the Block 10 mill. Although the results obtained cannot be regarded as being complete, in that only limited quantities of material were treated, and that the rolls were kept running at the same number of revolutions throughout, nevertheless, sufficient guidance was given to show that, with the several classes of ore, it would be inadvisable to alter the present method of crushing. From the data in Table I., obtained from trials on three separate lots of material, it will be seen that the amount of slimes and meal produced by the rolls in the worst example of single-stage reduction was only 5 per cent. greater than that produced from the same class of material in series-reduction, while the power consumed was in favor of the single-stage reduction to the amount of 29 per cent. Although the

roll-speed remained the same throughout the series trials, the opening between the roll-faces was altered to suit conditions as the reduction progressed.

TABLE I.—*Comparison of Results of Single-Stage and Series-Reduction.*

	Series-Reduction. Weight Per Cent.			Single-Stage Reduction. Weight Per Cent.		
Slimes and meal in ore from storage-bins,	2.89	6.08	3.6	2.93	6.28	3.28
Produced by rolls,	12.29	7.89	14.88	14.59	12.97	19.89
Loss—probably slimes,	4.80	5.2	4.73	3.34	3.31	4.06
Horse-power consumed, in h.p. hours,	2.29	2.3	2.14	1.72	1.5	1.52

Trommels.—The two trommels which receive the crushed material from each set of rolls are cylindrical in shape, set parallel to each other, with an inclination of 8° from the horizontal, and are 5 ft. 2 in. long and 22 in. in diameter. The screens are punched with round holes ⅛ in. in diameter, and the trommels are chain-driven at 20 rev. per min. Water is sprayed on to the outside of the screens by means of perforated pipes set parallel to them.

Until quite recently the work done by the trommels in the Block 10 mill was far from satisfactory, as sizing-tests showed that the material delivered from the trommel-ends contained about 30 per cent. of fines, which should have gone through the screen. Each trommel now has four pieces of 1½- by 1½- by ¼-in. angle-iron bolted longitudinally to the inside of the frame at equal distances from one another. The action of these angle-irons carries the crushed ore up to a certain height and lets it fall back with considerable impact, instead of having it carried up simply by the friction of the screens, from which it quietly slips downwards after having been lifted a comparatively small distance. The introduction of angle-irons has much improved the work, and the quantity of undersize wrongly carried over the ends of the trommels has been considerably reduced.

Raff-Wheels.—The oversize from the trommels is delivered to raff-wheels, which are about 14 ft. 6 in. in diameter, and 12 in. wide at the rim. The material falls into the inner circumference of the wheel, on which pockets are formed by joining the flanges of the rim by strips of iron placed at such an angle across the circumference as to carry the raffs to the full

height of the wheel, and deposit them on an apron, down which they travel to the rolls. The hub and the rim of the raff-wheel are of cast-iron. The spokes or arms are of timber, and support the rim from one side; tongued and grooved 0.75-in. flooring-boards are then screwed to the arms, making a water-tight diaphragm, which extends from the hub to the rim; a lining of ⅛-in. steel plates is then bolted on to the inside of this diaphragm, and extends round the circumference for a width of about 2 ft. 6 in., and is placed there to take up the wear of the particles delivered from the trommels.

In the Block 10 mill each raff-wheel is keyed to one of the roll-shafts, and acts as a fly-wheel, thus assisting the work of the rolls. One great disadvantage of this arrangement is that the peripheral speed of the wheel is so great that there is a tendency to overthrow the raffs, which then fall to the floor below instead of upon the apron. A speed of 12 rev. per min. is quite fast enough for a raff-wheel 14 ft. 6 in. in diameter, whereas, through being directly attached to the roll-shafts, the number of revolutions is actually 15. Another disadvantage is that the rolls cannot be speeded up without further interfering with the efficiency of the raff-wheel. Raff-wheels, though expensive in first cost, are very economical in practice. A trial between an elevator and a raff-wheel, working in the Block 10 old mill under exactly similar conditions, showed the latter to possess very great advantages. On account of the room which raff-wheels occupy it is not possible to place them within buildings except for only moderate lifts; their usefulness is, therefore, very much limited.

Hydraulic Classifiers.—A classifier, placed at the head of each jig, consists of an inverted cast-iron cone surrounded by a circular cast-iron launder, attached to the outside of the inverted base; there are two spigot-outlets near the inverted apex. There is also a third opening below these spigots, which is used for clearing away any obstructions which may hinder the flow through the outlets; but this opening is blocked up by a wooden plug while the classifier is in action. The three openings are contained in a casting of special construction, bolted on to the lowest part of the classifier. Classification is effected by means of a rising stream of water, which is introduced into the lower or narrower part of the cone by a pipe either passing

through the side of the cone or placed vertically in the center of the classifier. The through size from the trommels and shaking-screen enters the top of the classifier with a considerable amount of water, and is met by a further supply of water delivered from the classifier-pipe. The flow from this pipe is so regulated that there is an even overflow round the full circumference into the annular launder. By regulating the supply of water and the size of the spigot-outlets the size of the particles sent out in the overflow may be controlled. The overflowing material is carried away to a similar kind of classifier prior to its treatment on Wilfley tables. The internal width of the cone-base is 28 in., and the depth from the top to the bottom opening is 35 in.

Jigs.—Jigs of many different makes have been tried from time to time in the Broken Hill mills, but two of Australian invention and manufacture are now almost exclusively used. These are the Hancock and the May. The former is of the movable- and the latter of the fixed-sieve type, but since the May has replaced the Hancock in the more modern mills, it will be the only jig discussed here.

Each double jig generally consists of 10 compartments—four on either side for jigging, the fifth for receiving the tailings. Each jigging-compartment carries its plunger and sieve separated by a wooden partition, and situated above a steel or iron hopper of the form of an inverted pyramid. In the jigs of more recent construction this pyramidal hopper or box is made common to the opposite hutches and plungers on each side of the jig, but a steel diaphragm centrally situated, and extending longitudinally from the top of the jig almost to the bottom of the cone, prevents any interference between the currents produced by the respective plungers. In such cases the material passing through the sieves on the corresponding sides of the jig goes to a common delivery at the bottom of the hopper. The wooden sides of the jig are carried vertically upwards above the hoppers for a height of 16 in., and are finished off by a beading 1 in. in thickness. The jig is supported on girders placed between the hoppers, and at either end. Four coarse and four fine jigs are in use.

The sieves, placed 6.5 in. below water-level at the head of the jig, have a slight fall towards the tailings end, the differ-

ence in level between the two ends generally being about 1 in. The sieve is supported on grids, which are in turn carried on iron bars let into the woodwork at the sides. On top of the sieve is placed another grid, with openings corresponding to those of the lower one, and the three are pressed hard together and screwed down to the carrying-bars by means of studs. All joints made by the grid are then securely calked with yarn or other material to prevent leakage.

The sizes of the hutches vary in the different mills, but are generally about 3 ft. 6 in. by 2 ft. 6 in. for the coarse jigs, and about 3 ft. 4 in. by 2 ft. for the fine. The sieves are made of brass woven-wire, the mesh being proportionate with the size of the material fed to the classifier. In the Block 10 mill the hutches of the jigs are:

	Coarse Jig. Mesh.	Fine Jig. Mesh.
No. 1 hutch,	8	10
No. 2 hutch,	6	10
No. 3 hutch,	6	8
No. 4 hutch,	5	6

There is a great variation in the practice of the different mills on account of the differences in the nature of the ores.

Ragging, generally consisting of cast-iron shot, is placed on the sieve, and, shortly after commencement of jigging-operations, becomes intimately mixed with the heavier ore-particles on their way to the hopper below. This mixture of ore and ragging is called the "jig bottom," and one of the first requirements for successful jigging is the keeping of a good bottom. The plungers receive their motion through arms which extend from oscillating shafting placed centrally and longitudinally above the jig. This shafting is divided into two parts, each of which operates four plungers.

The plunger-body, of the same length as the hutch, is about 14 or 15 in. wide in the coarse jigs, and about 12 in. wide in the fine. It is composed of a cast-iron frame surrounded by a rubbing-board, which is bolted to it. The holes in the frame are slotted to allow of the widening of the rubbing-boards as these become worn. Attached loosely to the bottom of the plunger is the clack, made of pine 1 in. thick. The office of the clack is to close entirely the clack-opening in the plunger during the downward stroke, so that a fast upward current of

water may be produced in the hutch, and to open on the upward stroke, to allow a more gentle downward flow to take place. The clack of a coarse-jig plunger is generally about 2 ft. 6 in. by 9 in., and the clack-opening about 2 ft. 6 in. by 6 in.; there is generally about 0.5 or $\frac{5}{8}$ in. clearance between the clack and the plunger.

The plungers of the coarse jigs make about 180 and of the fine jigs about 200 pulsations per minute; from 1.5 to 2 h.p. and from 1 to 1.5 h.p. are respectively required for producing the motion.

When jigging is in operation, the products of the first two hutches on either side are concentrates; those of the next two hutches on either side are middlings; the material passing over the fourth hutches and falling into the fifth hoppers is tailings. The destination of these products has already been explained; they are carried thither by means of launders, which receive their supplies from the spigot placed in the bottom of each hopper. The spigots have openings varying from 0.5 to 0.75 in. in diameter, and deliver mixed solids and water in continuous streams. The capacity of the jig will vary with the nature of the crushed ore fed to it. If the feed consists of free galena of fair size, mixed with light quartz, a good throw may be given to the plungers, and a large capacity may be obtained, but with fine-grained galena mixed with zinc-blende, quartz, and rhodonite, the result is far different. Under ordinary circumstances, a coarse jig may treat from 5 to 7 tons per hour, and a fine jig from 4 to 5 tons per hour. Much water is fed through the classifier-spigots with the ore, but a further supply is generally necessary, and is drawn from a pipe placed at the head of the jig. The amount of make-up water will depend upon the quantity entering with the jigging-material, and that leaving through the outlet-spigots; it is therefore difficult to give even approximate figures. A rough estimate may be taken of 5,500 gal. per hour for coarse jigs and 2,200 for fine jigs.

In the Block 10 mill it is the practice, as far as possible, to produce concentrates from the coarse jigs containing 65 per cent. of lead, and from the fine jigs, 55 per cent. of lead. The following working-examples are given of the value of the material leaving each hutch. It should be remembered that the tailings from the coarse jigs are passed over a shaking-screen

before going to the dump, and that the tailings from the fine jigs pass to the grinding-pans. In the example given the bulk assays from the two concentrates-hoppers on the coarse jigs amounted to 65.2 per cent. of lead, and on the fine jigs, 55 per cent.

Lead-Content of Material from Hutch.

	Coarse. Per Cent.	Fine. Per Cent.
Hopper 1,	67.0	59.0
Hopper 2,	62.8	51.0
Hopper 3,	28.7	21.5
Hopper 4,	13.3	12.3
Tailings,	3.8	6.2

In order to exemplify the fine-grained nature of the galena which is being sent out as concentrates in this mill, the sizing-results obtained from average samples are:

	Coarse Jigs. Weight.	Coarse Jigs. Lead.	Fine Jigs. Weight.	Fine Jigs. Lead.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
On 20-mesh,	12.7	52.0	0.3	38.0
On 40-mesh,	26.2	74.0	26.7	45.9
On 60-mesh,	11.3	77.6	19.4	50.4
On 80-mesh,	23.6	72.4	33.0	56.3
On 100-mesh,	......	......	......	......
Through 100-mesh,	26.2	52.8	20.6	54.1
	100.0	65.7	100.0	51.8

Classification.—Metallurgists will no doubt draw attention to the absence of extensive classification of the crushed material before it is fed to the jigs. In many other mining-fields elaborate arrangements are made for feed-classification, whereas in Broken Hill the only treatment preliminary to jigging is the partial washing-out of the meal and slimes. Very complete trials and experiments have been carried out at various times for the purpose of discovering whether jigging could be improved by classifying closely, but, for some reason which has not been explained, it has always been found that the best results are obtained from the system as now in use. The extent to which classification takes place under ordinary working-conditions in the Block 10 mill is shown in Table II.

Elevators.—Elevators are of the belt type. The framework of the structure has a slope of from 20° to 25° from the vertical,

TABLE II.—*Classification at Block* 10 *Mill.*

	Coarse Jig Feed.				Fine Jig Feed.			
	Before Classification.		After Classification.		Before Classification.		After Classification.	
	Weight.	Lead.	Weight.	Lead.	Weight.	Lead.	Weight.	Lead.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
On 20-mesh............	43.2	13.6	42.5	12.0	9.8	14.4	13.0	13.0
On 40-mesh............	21.4	15.6	26.7	13.8	42.0	15.8	44.4	16.0
On 60-mesh............	8.0	15.4	9.0	17.6	14.7	19.2	15.1	21.6
On 80-mesh............	8.8	16.7	10.8	19.5	21.4	29.0	18.7	26.0
On 100-mesh...........	......	......	......	......	......	......	......	......
Through 100-mesh.	18.6	18.5	11.0	22.5	12.1	30.0	8.8	34.2
	100.0	15.35	100.0	14.94	100.0	20.7	100.0	19.92

and in some cases is more than 40 ft. in height. When material has to be raised through heights of 60 ft. or more it is preferable to make use of two separate elevators. The belt and buckets receive their motion through gear-wheels attached to the shafting of the top pulley and travel at a speed of about 250 ft. per minute; the gear-wheels are belt-driven.

Ball-Mills.—Ball-mills are machines for reducing the size of the material sent to them, and the action depends on the impact of steel balls placed, together with the ore, within a revolving cylinder or drum. Ball-mills vary in size and capacity; the following figures apply to a No. 3 Krupp mill, in which the drum, 5 ft. 3 in. in diameter, is attached to a shaft, 9 ft. long and 5 in. in diameter, through the medium of a cast-iron sleeve or center. This hollow center entirely surrounds and protects that part of the shafting which is within the drum. The casting of the. center is enlarged at one end in order to carry the feeding arrangement, which consists of helical passages communicating from the outside to the interior of the drum. This form of construction not only prevents the egress of the balls, but assists the feeding-in of the ore by acting as a short screw-conveyor. Cast-iron side-plates or drum-ends, 5 ft. 3 in. in external diameter and 1.25 in. thick, are bolted to either extremity of the center, and wearing-plates of hardened steel are attached to them to take up the wear due to the balls and material. One drum-end carries a manhole-opening and door through which access is gained to the interior of the mill. The circumference of the drum is composed

of hardened-steel castings bolted between the two drum-ends or side-plates. These outer plates do not form a true circle, but are so constructed that there is a step between the end of one and the beginning of the other; by this means the balls and material fall through a height of about 6 in. in leaving each successive plate as the drum revolves. There is an opening of about 2 in. between the bottom of one outer plate and the top of the next, where they overlap. All the outer plates are perforated with holes $\frac{3}{16}$ in. in diameter, through which much material passes with each revolution. Studded on to the external circumferential flanges of the outer plates are the inner screens, punched with holes $\frac{3}{32}$ in. in diameter. These screens do not cover the whole circumference, but have openings between them; they are bent and lapped so as to form baffle-plates to lead that material which fails to pass through the outer screens back into the ball-mill for further reduction. The outer screens, of steel sheets punched with $\frac{3}{64}$-in. round holes, are attached to the periphery of the drum-ends by means of bolts passing through flanges on the latter. The whole mill is inclosed in a steel casing, $\frac{1}{8}$ in. thick, which has suitable openings for access to the working-parts and to the perforated pipes which spray water on to the outer screens. The drum is revolved at from 28 to 30 rev. per min. by belt-driven gear-wheels, and requires from 8 to 10 h.p. for driving purposes in treating about 4 or 5 tons of material per hour.

While the ball-mill is in operation a portion of the material during each revolution finds its way through the openings in the wearing-plates, through the inner screen, and through the outer screen, while that too large to pass through either of the two latter is returned to the interior of the mill through the openings constructed for the purpose, as already described. The crushing-balls are 5 in. in diameter when new, but gradually become reduced in size by the wear which goes on during the crushing-operation; they are removed when reduced to about $1\frac{1}{2}$ or 2 in. in diameter. In running continuously, fresh balls to the weight of about 85 lb. are added weekly to take up this wear. The cost of up-keep due to the wear and tear on balls, and the ball-mills themselves, is heavy, but they are very economical otherwise on account of the small power required for their operation. As reducing machines they are entirely

satisfactory in dealing with material such as that shown in the examples below, but for very fine material they are not to be recommended. The size of the material leaving the ball-mill is entirely governed by the mesh of the outer screen, and as long as the screens are kept in good repair no oversize beyond that required will leave the mill.

The following are examples of the feed to, and the crushed product from, a ball-mill of the above description:

	To Ball-Mill.		From Ball-Mill.	
	Weight.	Lead.	Weight.	Lead.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
On 20-mesh,	34.7	15.8	9.8	14.4
On 40-mesh,	34.8	19.8	42.0	15.8
On 60-mesh,	9.1	27.2	14.7	19.2
On 80-mesh,	11.3	27.0	21.4	29.0
On 100-mesh,	......	......	......	......
Through 100-mesh,	10.1	34.0	12.1	30.0
	100.0	21.33	100.0	20.7

Grinding-Pan.—The majority of pans in use in the Broken Hill mills are 5 ft. in diameter and about 2 ft. 6 in. deep. The pan proper is made of cast-iron, and is so constructed that a hollow cylindrical column stands up in the center and allows of the passage of a vertical shaft used for operating the grinding-apparatus. To the bottom of the pan are fixed dies of hard iron or steel about 3.75 in. thick, so placed that there is a radial opening between the adjoining dies about 2 in. wide. The dies are so shaped as to give a curved direction to the radial openings between them. The dies are considerably shorter than the radius of the pan, and are set about 2 in. clear of the circumference and of the central column. A vertical shaft passes through the central column, and is supported in a bearing containing the usual hardened-steel foot-step. This shaft is made to revolve through the agency of bevel-gear operated by shafting and pulley placed below the pan. The pan and all its belongings are supported on an iron framework about 2 ft. 6 in. high. The material to be ground is fed into the pan as close to the center as possible, whence it finds its way through the shoes and dies to the outside circumference. The finest material then rises to the top of the swirling water, and sufficient extra water is added to carry it over the lip. Such material

as is still too heavy to be lifted over the lip by the current of the escaping water is drawn between the shoes and dies through the curved openings for further comminution. By regulating the water-supply the fineness of the grinding is therefore also regulated.

In each pan are 18 shoes and dies, weighing about 1.75 tons. As the shoes and dies become worn, annular cast-iron compensating-rings are placed on top of the rotating shoes in order to make up the lost capacity due to the lessened weight. The amount of power required to operate these pans will vary with the weight of the compensating-rings imposed. The capacity will vary with this weight, with the hardness of the material to be ground, with the size of the particles entering and the fineness to which the particles must be ground before leaving. In the example given below the output is about 1 ton an hour, and the pan uses 10 h.p.; the material is very hard. The life of a set of hard-iron shoes and dies working under these conditions is about 14 weeks.

The following is an example of regrinding in the Block 10 mill:

	Fine Jig Tailings.			
	Before Grinding.		After Grinding.	
	Weight.	Lead.	Weight.	Lead.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
On 20-mesh,	9.7	5.7	......	......
On 40-mesh,	39.4	5.6	1.6	2.0
On 60-mesh,	13.9	5.4	14.2	3.0
On 80-mesh,	21.5	5.3	51.4	3.2
On 100-mesh,	......	......	......	......
Through 100-mesh,	15.5	13.8	32.8	13.8
	100.0	6.78	100.0	6.63

Spitzkasten.—A *spitzkasten* used to settle and classify slimes is a series of inverted cones of gradually increasing size. The slimy water enters its narrow and shallow end, and travels towards the broader end with ever-decreasing velocity. By this means the heaviest particles in suspension are deposited earliest and the lightest last. The thickened material is drawn from the various compartments by means of spigots. *Spitzkasten* vary in dimensions, those having the largest surface-area being used for the slimes most difficult to settle. The following details are taken from a *spitzkasten* of three compartments

used for settling fine slimes: Inclination of keel, 8°; width of inlet, 5 ft.; width of outlet, 7 ft.; depths, 2 ft. and 3 ft. 6 in.; length from inlet to outlet, 15 ft. The whole is composed of Oregon pine, and the keel is 10 by 6 in., the sheeting 10 by 2 in., and the ribs and ties 6 by 4 in. For coarser material *spitzkasten* having four compartments, and the following dimensions, are used: Length of top and bottom, 15 ft. and 10 ft.; depth, 2 ft. 1 in. and 4 ft. 8 in.; breadth, 2 ft. 8 in. and 7 ft. 1 in.

The following is an example of the classification effected by a four-compartment *spitzkasten*:

	First. Compartment. Per Cent.	Second. Compartment. Per Cent.	Third. Compartment. Per Cent.	Fourth. Compartment. Per Cent.
On 20-mesh,	6.0	......	......	......
On 40-mesh,	20.7	2.6	......	......
On 60-mesh,	15.7	2.7	......	......
On 80-mesh,	27.8	38.5	......	......
On 100-mesh,	......	......	......	......
On 150-mesh,	26.8	43.5	1.7	1.7
Through 150-mesh,	3.0	12.9	98.3	93.3
	100.0	100.2	100.0	100.0

It is noticeable that all material fine enough to pass through 80-mesh will pass through 100-mesh. Similar results have been obtained from the majority of the sizing-tests made on the various products of the Block 10 mill.

Slime-Washing.—Dressing-machines of three different makes are used in the Block 10 mill for washing the fine sand and slimes; these are Wilfley tables, Card tables, and Warren vanners. Wilfley tables have become so well known by their widely-distributed use that a detailed description is unnecessary. The motion of the Card table is similar to that of the Wilfley, but there is a difference in design in the head-gear which produces it. There are also several differences in the dressing-surfaces, in the methods of support, and in the minor details. In both classes of machine the table-top is constructed of timber strengthened with steel, but whereas, in the Wilfley the dressing-surface is composed of linoleum with raised longitudinal riffles, in the Card the dressing-surface is of pine, in which longitudinal grooves are cut. The capacity and consumption of dressing-water of these tables will vary with the

nature of the material fed to them. The Warren vanner is of the belt description, and in some respects resembles both the Triumph and the Lührig. It makes longitudinal vibrations, 0.75 in. in length, at the rate of 220 per minute, has a lateral slope to the belt of about 4°, and the belt travels forward at the rate of 12 ft. per minute. The surface-area of the upper or dressing-side of the belt is 12 by 4 ft.; 20 Card tables, 8 Wilfley tables, and 22 Warren vanners are in use in Block 10 mill.

The following example of the work done by Wilfley tables is taken from actual practice. For the purpose of gaining detailed information as to the quality of the work being done by each part of the table, special samples were taken, as work progressed, from succeeding sections of the discharge-side, as here shown:

	Length of Division. Feet.	Quantity Per Hour. Pounds.	Weight. Per Cent.	Lead. Per Cent.
Crude material,	......	805	......	17.3
Concentrates,	2	130	16.2	67.0
Middlings,	2	207	25.7	9.4
Tailings 1,	2	89	11.0	4.0
Tailings 2,	2	116	14.4	1.5
Tailings 3,	2	47	5.8	3.2
Tailings 4,	2	28	3.5	6.3
Middlings 5,	2	188	23.4	14.4

Slime-Settlers.—These wooden "V"-shaped settlers have a keel made of 10 by 6 in. Oregon pine, resting on the sole-pieces of the rectangular frame, which contain the settler proper. The framing, which is composed of 6 by 4 in. Oregon pine, strained together by vertical and horizontal 0.5-in. bolts, has a width, inside timbers, of 8 ft. and a height of 4 ft. 9 in., also inside timbers. These frames are placed 5 ft. apart. Stretched from the keel to the vertical side of the framing are ribs, 6 by 4 in., set at an angle of about 50° from the horizontal, and the sides of the settler are formed of 10- by 2-in. planks, nailed on to these ribs. The water is delivered along the whole of one side of the settler, and overflows throughout the whole length of the opposite side. Openings are placed at intervals along the sides of the settlers, close to the keel, and are closed by doors 8 by 6 in. in dimensions. The doors which cover these openings are of cast-iron, and are moved up and down by means of a rack and pinion in a cast-iron frame, which also contains a cast-iron lip. The removal of slime

through these doors is much more economical and expeditious than was the case with the old form of settlers; better clarification also takes place and much less water is lost. The surface of all settlers and *spitzkasten* contained in the mill, with the exception of bins and service-tanks, amounts to 5,700 sq. feet.

Water-Storage.—Three large service-tanks, 20 ft. in diameter, and each having a capacity of 28,000 gal., are placed on the rising ground at the back of the mill; the water from them gravitates to all parts of the plant. Another tank, with a capacity of 6,000 gal., is placed on a lower level at the side of the mill, and supplies the wash-water for the slime-tables and vanners. The water delivered into these tanks consists of the clarified overflow from the slime-settlers, from which it is elevated by centrifugal pumps. Water is also supplied to these tanks from the underground department and from the local water-supply. As the clarified water still retains a certain proportion of solids in suspension, adequate means are provided for cleaning out these tanks periodically.

Pumps.—With the exception of two three-throw plunger-pumps placed at the foot of the dumps, centrifugal pumps are used throughout the mill for raising the clarified water and the water containing slimes and grit. Pumps of fairly-good efficiency are used for dealing with the clarified water; those dealing with the water containing slimes and meal are of lower efficiency, but are capable of withstanding much greater wear and tear. The amount of solids contained in the water elevated by these pumps varies from 0.5 to 1 lb. to the gallon. The proportionate sizes of the solids may be judged from the following data:

	No. 1 Pump.		No. 2 Pump.		No. 3 Pump.	
	Weight.	Lead.	Weight.	Lead.	Weight.	Lead.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
On 20-mesh,	2.3	6.2	3.7	9.1	......	......
On 40-mesh,	5.5	6.0	9.7	8.3	......	......
On 60-mesh,	3.3	5.9	6.2	7.9	1.1	8.3
On 80-mesh,	14.3	6.0	25.9	7.3	4.2	5.0
On 100-mesh,	......	......	......	......	......	......
On 150-mesh,	3.6	6.2	6.7	9.4	3.5	4.5
Through 150-mesh,	71.0	22.0	47.8	20.1	91.2	18.9
	100.0	17.3	100.0	13.76	100.0	17.65

There are now 10 pumps in use in the mill, which handle 216,000 gal. of water per hour, but, as some of the pumping is

done in stages, this figure does not actually represent the quantity of clarified and slimy water in circulation. There is a loss of from 105 to 110 gal. of water for every ton of ore treated.

Power.—The use of electricity for driving the entire milling-plant was first introduced into Broken Hill by the Block 10 Co., and, being the pioneers, many initial difficulties were encountered.

Recent additions to the mill have greatly increased the consumption of power, and have necessitated the installation of additional pumping-appliances. For the months of June, July, and August, 1906, the average weekly quantity of electric power supplied to the mill amounted to 29,000 kw., while for the corresponding period of the year 1908 this had been increased to 42,000 kilowatts.

The following is the average horse-power required by the various machines in working under normal conditions. It will be observed that the total horse-power per hour amounts to 448; under certain conditions this may be raised to as high as 500.

	Average Horse-Power.
Rolls (raff-wheels, shakers, etc.),	120
Coarse jigs, fine jigs, and eight elevators,	40
Ball-mills,	40
Grinding-pans,	80
Vanners, tables, and elevator,	34
Clear-water pumps,	104
Slime-pumps,	30
Total,	448

The Austin crushers and the aërial tram are on a different electric circuit from the mill, and absorb from 62 to 68 h.p. per hour.

Proportion of Concentrates.—A great change has taken place in the nature of the ore raised from the mine during the past two years. The presence of much greater quantities of zinc-blende and the finer-grained and denser description of the material obtained from the lowest levels have rendered fine reduction necessary on a much greater proportion of the ore milled than was formerly the case. The mill has been changed to suit the altered circumstances by adding extra slime-dressing appliances to deal with the increased production of slime, and also by installing grinding-pans and additional tables for the re-

treatment of the tailings from the fine jigs. The magnitude of the change in the ore may to some extent be gauged by the fact that where from 60 to 70 per cent. of the concentrates produced were previously obtained from the coarse jigs, only about 53 per cent. are now obtained from that source. The following are the proportions of concentrates being produced from the various departments of the mill at the present time:

	Weight. Per Cent.	Lead. Per Cent.
Coarse jigs,	53.1	63.0
Fine jigs,	19.2	53.4
Tables and vanners,	23.1	56.7
Regrinding,	4.6	60.9
	100.0	60.6

The mill treats 3,000 tons of ore per week.

Sampling.—Very complete arrangements are made for the proper sampling of the mill-products and by-products, as well as of the crude ore. The sampling is in charge of the assay officers, and is therefore independent of interference by the concentration-department. The crude ore is sampled every hour as it is delivered from the bins by the feed-rollers, and a check-sample is also taken at the intermediate half-hours. Samples of all products and by-products are taken at regular intervals throughout the mill and on the dumps during the whole of each 8-hr. shift. These samples are then sent to the assay-office. To make sure of the correctness of the sampling, another officer of the assay-department goes over all the dumps once a day and takes check-samples representative of the work done during the previous 24 hr. At 11.30 a.m. each day the results of the previous day's concentration are posted in the mill for the information of all employees concerned, special attention being drawn to any faults which may have occurred in the quality of the work done. Earlier in the morning the mill superintendent receives an advance sheet showing the most important assays, and these serve as a guide to him until the arrival of the full and detailed list.

Under the present contract, the concentrates are sold on trucks at Broken Hill, and shipping-samples are taken for the purpose of determining the assay-value and moisture contained in the concentrates shortly before the railway-trucks are dis-

patched to the weighing-yards. As freights are heavy, it is usual to store the concentrates in the dispatch-bin for a few days to allow them to drain before being loaded. The moisture generally amounts to 4.5 per cent. of the weight of the loaded concentrates when the trucks are dispatched.

Recovery of Metals.—The recovery made from the silver-lead concentration-mills on the Broken Hill field is generally understood to refer to only that proportion of the metals which leaves the mills in the form of leady concentrates. Credit is not taken for the amount of metals contained in the slimes and in the by-products heavy in zinc, which must go through another process of concentration before they are ready for the smelter. The only correct way in which to calculate recoveries is from assays made of representative samples of the untreated ore, and of the resulting concentrate. Recent experiments made on the Broken Hill by-products have not only recovered from 70 to 80 per cent. of the zinc in the form of a marketable zinc-blende, but have also recovered even higher proportions of the lead- and silver-contents. The figures published by several of the Broken Hill companies show recoveries obtained from the ordinary lead-concentration processes of from 60 to 75 per cent. of lead, and from 37 to 55 per cent. of the silver-content of the crude ore.

Costs.—In Broken Hill the ore treated by the various mills is so different that no fair comparison can be made with regard to either the costs of concentration or the recoveries obtained. The wear and tear on the machinery dealing with a fairly-soft, free-milling ore, will be much less than on that dealing with a harder and more refractory material. Also, more labor and further treatment will be required with the latter than with the former.

Data appearing in the periodical reports of several of the companies show recent milling-costs to vary from 4s. 8½d. to 7s. 1½d. per ton of crude ore treated. There has been a considerable rise in the costs of all departments of mining during recent months, and the figures just quoted include increased items of expenditure due to the higher rate of wages now existing and to the increased cost of water. Power-costs are also heavy in Broken Hill, since water for steam-making is charged at the rate of 5s. per 1,000 gal., and coal, which has 250 miles

of land-carriage from the nearest port, costs from 30s. to 35s. per ton by the time it is delivered on the mines.

Accurate and detailed costs of every milling-department on the Block 10 works are kept, and these include charges for power, up-keep, labor, etc. The heaviest expenditure is on account of the daily wages paid for labor in milling and in the up-keep and repair of the machinery. The mill employees receive the following rates per 8-hr. shift, and are paid 25 per cent. extra for all overtime: Truckers and laborers, 8s. 7½d.; jig-men, 9s. 6d.; vanner-men, 9s. 1d.; boys, 5s. 9d.

The following is a summary taken from the detailed expenditure, as shown on a recent cost-sheet extending over a period of one month:

	s.	d.
Daily wages,	2	11.78
Power,	1	4.49
Water and stores,	1	1.28
Miscellaneous,	0	0.60
	5	6.15

To enable the mill superintendent to exercise a proper care over his expenditure, he is placed in possession of returns showing the detailed costs of his work at regular intervals. His cost-sheets show not only the amount expended in labor, stores, power, etc., for each section of his work, such as crushers, rolls, and ball-mills, but, for the purpose of comparison with previous expenditure, each separate item is carried out to fractious of pence per ton of crude ore treated. Included in the same statement are figures relating to the tonnage, assay-value, and mineral-content of the crude ore treated, and of the concentrates produced, as well as to the recovery obtained. It takes but a glance to show him in the instance quoted that he has treated, say, 12,000 tons of ore, has produced 2,000 tons of concentrates, has obtained a recovery of 68 per cent. of lead, and that his total cost is 5s. 6.15d. per ton of crude ore.

Sale of Concentrates.—Concentrates are generally sold on trucks at the mine or at the smelting-works or sea-board, and the weights as shown by the railway weigh-bridges are taken as the basis for all sales. Before the trucks go to the weighbridge, samples from each one are taken in the presence of the buyer and seller or their agents, and from these samples the

amount of moisture contained in the concentrates, and their assay-value, are determined. For the purpose of assaying, it is usual to cut the bulk-sample down and divide it into three equal portions, one of which goes to the buyer, one to the seller, and the third is sealed up and carries the seals of the buyer and seller. The first two portions are duly assayed by their respective holders, who afterwards come together and compare the results obtained. If these agree within one unit of lead or zinc and one ounce of silver, then the assay-value of the shipment is taken to be the mean of the assays. If, however, there is a greater difference than that mentioned, the sealed packet is sent to an umpire mutually agreed upon, and that of the three assays which is intermediate between the other two is accepted as being the correct one.

The silver- and lead-content are paid for at London quotations, averaged over a certain period, say, three months, after deductions have been made for moisture and smelting-charges; fines are also imposed to the extent of so much per unit per ton for every unit of zinc over 10 per cent. Contracts for the sale of concentrates are made for extended periods, and in these contracts certain definite smelting- or returning-charges are specified.

Conclusion.

The concentration of silver-lead ores covers such a wide range that it has been possible, within the limits of this treatise, to deal with only the most important points and to touch upon them but slightly. Many of the statements made might, no doubt, be open to contradiction if applied to other districts, or even perhaps to the other mills on the Broken Hill field, but, knowing the great differences which occur in the various classes of ore, not only in the one limited field, but in the one mine, I have, as far as possible, given figures obtained only in actual practice. Although these figures do not always show perfect accord, it has been adjudged wiser to make use of results obtained under working-conditions rather than to build up statements partly or almost entirely based on theory and supposition. It is freely admitted that much of the work set forth could be materially improved, but many of the faults lie not so much with the system adopted as with the lack of care or skill or both on the part of the individuals employed. With

an ore which varies greatly in the different parts of the underground workings, sudden changes must be expected in the quality of the mill-feed, and these changes must, for the time being, even when the utmost care is being exercised by the most skilled workmen, produce a deleterious effect on the mill-recoveries. Experimental work is always in hand with a view to increasing the efficiency of the milling, and no trouble is spared in taking samples and making assays for the purposes of discovering the causes of faulty work.

It must always be remembered that concentration is a commercial undertaking, and is governed by monetary considerations. If a mill were worked entirely for the sake of science, high recoveries could be obtained, although at a highly-increased working-cost. If the work is to be carried out on commercial lines, the monetary value of the product to be obtained from each proposed addition to a plant, as well as the cost of obtaining that product, must be carefully computed, and if, finally, the addition is constructed, careful observations must be made in order to prove whether the estimated results have been realized. It is the constant aim of every metallurgist engaged in the concentration of silver-lead ores to improve his milling-work, but, as great fluctuations occur in the prices of the valuable metals with which he has to deal, any heavy expenditure for the proposed improvement of his returns must be approached with due caution.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Treatment of Slime on Vanners.

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(New Haven Meeting, February, 1909.)

SOME time ago the Detroit Copper Mining Co. had to decide the question whether it would pay to re-treat slime-tailings, and several machines were tested in order to ascertain the type of construction which would give the greatest saving. In previous tests on the ore of this company, several tables proved far inferior to the Frue vanner, and for this reason they did not receive any serious consideration. In the late tests, however, one table was found to surpass the vanner in the percentage of saving made.

The Frue vanner used in these tests had been adjusted approximately to the slope which the slime-vanners in the concentrator of the Detroit Copper Co. had at that time, about 2 in. between the inside of the wooden posts, or 0.272 in. per ft. It had not been determined, however, if this slope was the most economical one for treating the slime-feed in the mill, or for re-treating the tailings from the slime-vanners, and I was instructed to ascertain by test-runs, under different conditions, the best adjustments for the work to be done.

In order to determine if a change in the adjustments of the machines would improve the results, it seemed sufficient to continue running the tests between the vanner and the table before mentioned, keeping the feed and all other conditions the same, but varying the adjustments of the vanner from run to run.

I employed a vanner-man of long experience to operate these machines, who expressed the opinion that a slope between $2\frac{3}{8}$ and 2.5 in. between the posts (0.323 and 0.340 in. per ft.) would give the best results.

Some runs made this way gave a considerable improvement. While for a slope of 2 in. (0.272 in. per ft.) the saving had been only 40.5 per cent. of that of the table, it was 49.2 per cent. for 2.5 in. (0.340 in. per ft.) slope. A decrease of

the slope to 1.5 in. (0.204 in. per ft.) decreased the saving to 32.7 per cent. of that of the table. So far, these tests had been made with a smooth belt and with 224 (1-in.) side-strokes per minute. A reduction in the number of side-strokes from 224 to 188 increased the saving several per cent., and an increase of the slope to 3 in. (0.408 in. per ft.) added 8 per cent. to the saving.

An old corrugated belt was then tried, starting with 2.5. in. slope (0.340 in. per ft.). The saving was 78 per cent. of the saving of the table; for 3.5 in. (0.476 in. per ft.) it was 97 per cent., and for 4.5 in. (0.612 in. per ft.) the vanner made a better saving than the table. A decrease in the number of side-strokes slightly increased the saving.

These results could be interpreted in two ways. It was possible that the pulp, which had once gone over a vanner with a somewhat flat smooth belt (all vanners in the concentrator had smooth belts at that time), was not fit for re-treatment on a machine with practically the same adjustments. To take out mineral that could not be saved on the first vanner, either the smooth belt might have to be set steep, or a corrugated belt might have to be used. On the other hand, there was a possibility that the vanners in the concentrator had been run too flat, and therefore did not save all that could be saved. In this case a steeper setting of the vanners would increase the saving and would possibly make a re-treatment of the tailings unnecessary.

Since smooth belts only were in use, the first work was to determine the best adjustments of a Frue vanner with a smooth belt for treating fine feed. The second, to ascertain if a corrugated belt will do better on slime-feed than a smooth belt. Of course, any such difference in the saving for different adjustments, as reported above, could not be expected for the regular slime-feed, from which a large amount of concentrate can be easily extracted by almost any machine, while on a pulp which has already gone over a concentrating-table once, one machine may have hardly any effect, while another may save a considerable amount of mineral.

Some tests with different adjustments (mainly different slopes) convinced me that on this feed a higher slope also increased the saving. Since, however, this view was diametrical to the former practice in the concentrator of the Detroit Copper Co., accord-

ing to which slime-feed was treated on belts set as flat as possible (with the idea that the slimes should get a very small chance to run off), my views found opposition, and it was decided to have a contest, in which an experienced millman ran a vanner according to the established practice, while I ran another machine on the same feed according to my ideas. Two vanners were set side by side and fed by slime-pulp from a revolving distributer, so as to send the same amount of feed of the same quality to the two smooth-belt vanners. Four runs were made, averaging about 7 hr. each, from which the concentrates were collected, weighed, and assayed. One time the contestants changed machines between two runs. On an average of the four tests, the machine with the higher slope saved 10.1 per cent. more copper than the machine with lesser slope. One other point brought out by the contest was that, in order to effect a high saving, a certain amount of water, probably in excess of what is generally used on slime-vanners, is required.

The results of this contest indicated that it is possible to save a considerable part of the copper that heretofore has been lost, and that it would probably pay to determine accurately and systematically the most economical adjustments of a vanner in treating this feed.

The right way to find the best adjustments of a vanner for a certain feed is to vary all the elements that can be varied, and to determine the saving resulting from different combinations of adjustments. Since there are several variations possible, this scheme requires a large number of tests.

The principal elements which can be varied on the Frue vanner are: (1) the slope of the belt, (2) the amount of dressing-water, (3) the number of side-strokes, and (4) the speed of the belt. If each of these elements has a certain value, the machine will for a certain quality and quantity of feed produce a certain grade of concentrate. In practice, however, the grade of the concentrate which has to be made on the vanners will be a given quantity. It will be stipulated, for instance, to make a grade with only 10 or 20 per cent. of silica. This condition makes the above-mentioned adjustments interdependent. For instance, with a given number of side-strokes per minute, and a given slope of the belt, the vanner-man can

either use a certain amount of dressing-water and regulate the speed of the belt, or he can give the belt a certain speed and regulate the dressing-water so as to produce the desired grade of concentrate.

In most of these experiments the former way was chosen—namely, a certain quantity of dressing-water per minute was used. For this purpose a funnel was constructed, Fig. 1, having the opening, *A*, closed by a wooden plug, *B*, with an inserted glass tube, *C*. At the wide end of the funnel a tube, *D*, branches off. The funnel is set on a nipple, *E*, screwed on the pipe, *F*, which carries the dressing-water to the water-distributing box of the vanner. The water enters the funnel

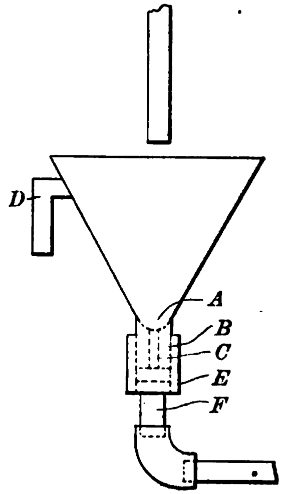


FIG. 1.—WATER-REGULATOR FOR VANNERS.

through its upper end. The inflow is so regulated that it fills the funnel and gives a small overflow through the pipe, *D*. As long as this small overflow is maintained the water running through the glass tube, *C*, is always under the same pressure, and the quantity passing through is constant. By changing the glass tube the quantity of dressing-water can be varied.

The quantity of feed was regulated in a similar way by passing the pulp through a short piece of iron pipe under a given head. In the later runs special efforts also were made to keep the consistency of the pulp uniform by using a separate settling-tank having a spigot-discharge controlled by a plug with an inserted iron tube, while the tank itself was supplied with the

same kind of pulp up to the settling-capacity for a clear-water overflow.

In order to get good results, it was found necessary to use a speed of the belt much more rapid than the ordinary adjustments allow. This was accomplished by replacing the cone-pulleys which regulate the belt-speed with larger ones.

The relative saving made on the two machines was determined by weighing, sampling, and assaying the concentrates produced in each run. Hand-samples of feed and tailings also were taken at regular intervals, but only the saving, based on weight and assay of the concentrates, was used for a comparison of the work of the machine under different conditions. In the present case this method of determining the saving is evidently the most reliable one. As a rule, the whole concentrate was dried and sampled so as to avoid errors due to faulty moisture-samples. At the start the concentrates were assayed by the cyanide method, but later these assays were used for preliminary work only, the calculations being based on electrolytic assays.

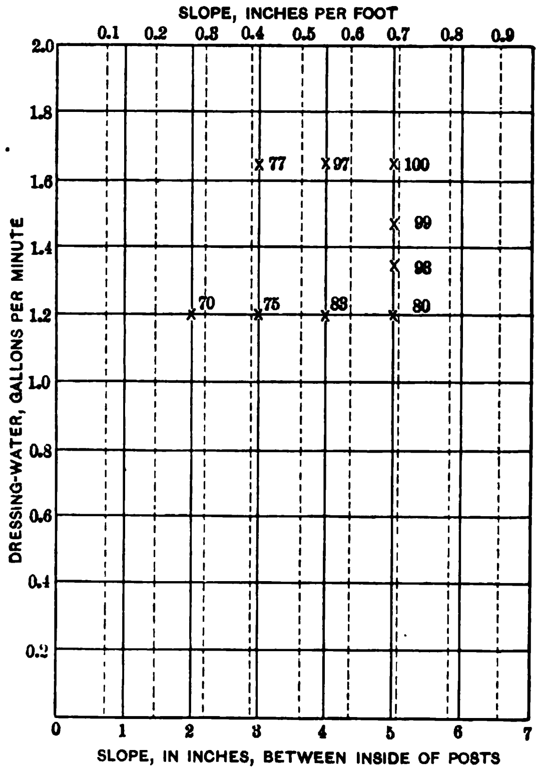
To eliminate errors due to the impossibility of keeping two machines in equally good condition all the time, the adjustments were reversed when practicable. For instance, when one machine had been run with a 3-in. slope and another with a 4-in. slope for one day, the next day the slopes were reversed, but since this called for frequent changes in both machines, which at times could not be easily effected, another method was generally followed. One machine, kept running in the same way for a whole series of runs, was called the standard vanner, and all the variations were made with the other machine. As an example, the results of one series of tests are given in Table I.

TABLE I.—*Effect of Variation of Slope.*

Test number		117 and 118	119	120	121	122	123	124	125
Slope.									
Between Posts.	In. per Ft.								
5 in.	0.680								
6 in.	0.816	....	......	....	....	....	118.9	116.7	
7 in.	0.952	113.2	......	....	....	114.2	....	....	114.8
8 in.	1.088	....	106	107.3	111.2				

The data in Table I. were obtained under the following conditions: dressing-water used, 2.7 gal. per min. (per 6-in. belt);

average quantity of ore treated per 24 hr., 8.95 tons; average amount of solid, 13.53 per cent.; average number of strokes per min., 188 (1-in. strokes); corrugated belt. The results are given in percentages of the results obtained by the standard vanner having a smooth belt, a 5-in. slope between posts (0.680



Smooth belt, 220 (1-in.) strokes per minute.
Feed: Lower vanner feed of D. C. M. Co. concentrator.
13.0 per cent. solid.
10.0 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.40 per cent. copper.
9.5 tons dry feed per 24 hours.
Saving is expressed in percentage of maximum saving obtained in this series of tests.

FIG. 2.—VARIATION OF SAVING WITH DRESSING-WATER AND SLOPE.

in. per ft.), using 1.66 gal. of dressing-water per min., and operated at 229 (1-in.) strokes per minute.

Fig. 2, showing the results obtained on a vanner with a smooth belt, expresses the saving in percentages of the maxi-

mum saving obtained in this series of tests. The figures represent this saving, the slope and the dressing-water being used as co-ordinates. This diagram shows that, in order to make a good saving, it is necessary to set a vanner much steeper than is done in common practice. The best saving was obtained with 5-in. slope between the inside of the posts (0.680 in. per ft.), and 1⅝ gal. of dressing-water per 6-in. belt. Under these conditions it was necessary to give the belt a speed of 120 in. per min., which is probably three times the rate of travel used in most mills, and far in excess of any speed that I have ever seen quoted. Both the large quantity of dressing-water and the high slope of the belt tend to produce a high speed. The results of these tests show that, in order to make a good saving on slime-feed with a vanner, it is necessary to give the belt a high speed, which can be done either by using a high slope or a large quantity of dressing-water. On a given feed the best combination can only be decided by experiment.

The details of Fig. 2 are not very complete, but since the determination of even these few figures took a long time, and since it was the expectation to discard the smooth belts should corrugated belts be found more economical, the data collected were considered sufficient.

Richards[1] mentions as extraordinarily high the rate at which the belts move forward in the Gates canvas-plant, operated at Jackson, Cal. At this plant very fine canvas-table concentrates are re-dressed on an end-shake vanner which has the extremely high slope of 1.5 in. per ft. I have tried repeatedly to use slopes approaching this one, but in every case with negative results, which may be due to the fact that the feed was not as fine or that an end-shake vanner will stand a higher slope. The amount of shaking-motion also is much larger in my tests, which may help to explain the difference. The shaking-motion is probably stronger than is practical in view of the difficulty of keeping the machines in good order, but as far as the saving is concerned, not much could be gained by reducing the strength of the motion, as will be seen later in this paper.

The question arises, why, if a high belt-speed gives so much better results, it is not used in some mills? Certainly, some

[1] *Ore-Dressing*, vol. ii., p. 660 (1903).

one must have tried high speed before, but I am inclined to think that no one ever investigated thoroughly the question of belt-speed. Most millmen trust a good deal to the eye, and a fast-traveling belt does not show the concentrate very plainly. Every one viewing two belts side by side, one at a fast speed and one at a slow speed, will feel certain that the slow-speed belt produces much more concentrate. The reason for this is, that in the case of the fast belt the concentrate is spread out over a larger surface in so thin a film that sometimes it is hardly perceptible. Other tests in the mills give misleading results. For instance, the belt which shows very little mineral behind the feed-box is frequently considered to make a good saving, but in some of our runs this test failed entirely. Belts showing mineral almost down to the tail-end often made much cleaner tailings and a better saving than did belts which looked clean over the entire length. Panning the tailings is more satisfactory, but the only safe way to determine the saving seems to be by actual assay.

One objection which has been raised against this way of running vanners is, that variation in the power would influence a fast-running belt more than a slow-running one. Whenever the driving-power loses speed a vanner begins to carry sand into the concentrates, on account of the gentler shaking-motion. And if a vanner be adjusted to the slower motion it will make too clean a concentrate as soon as the power recuperates, which means a loss of mineral, unless the vanner-man immediately readjusts the machine to the changed condition.

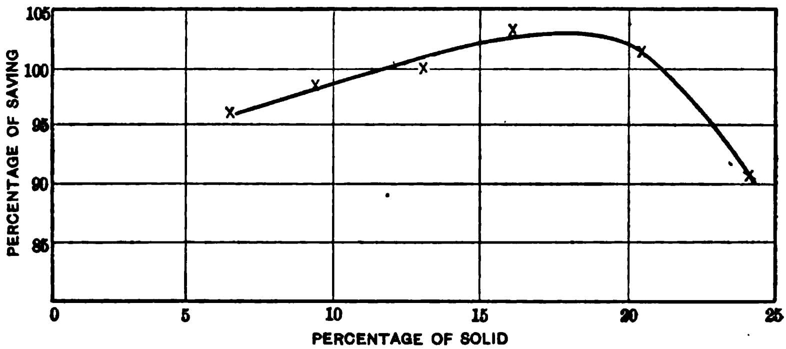
In other words, that losses will be heavier on a fast-traveling belt, on account of the higher slope. It has been found by experiment, however, that a machine with a high slope requires less regulation to meet changed conditions of power than a machine with low slope, and from this fact I infer that the losses due to lack of attention to the changes in power will be reduced, and not increased, by giving the vanners a high belt-speed. This condition, therefore, recommends a high belt-speed, particularly in places where the power is not very constant.

It may also be mentioned that a violent shaking-motion makes a vanner more independent of changes in the power, so that for plants having poor power it is advisable to use

a rapid belt-speed and shaking-motion. While high speed will cause a belt to wear out faster than a slow speed, the results obtained at the concentrator of the Detroit Copper Co. show that the cost of increased wear will be made up many times by the increased saving.

The feed treated on the vanners in the tests shown in Fig. 2 averaged 13 per cent. of solid. Screen-analyses made at different times showed between 6 and 14 per cent. of residue on a 200-mesh screen of the Denver Fire Clay Co. The average load was 9.5 tons (dry) per 24 hours.

We also investigated the consistency of pulp best suited for



Smooth belt, 209 (1-in.) strokes per minute, 2.20 gal. dressing-water per minute, 4-in. slope between posts (0.544 in. per foot).
Feed: Lower vanner feed of D. C. M. Co. concentrator.
9 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.48 per cent. copper in feed.
7.93 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made with a pulp of 12 per cent. solid.

FIG. 3.—VARIATION OF SAVING WITH PERCENTAGE OF SOLIDS IN FEED.

treatment on vanners. The experiments made were similar to those described above, except that the feed was thickened in a settling-tank with clear-water overflow before it reached the distributer. Alternating daily, extra water was applied to one of the two machines, and in this manner the saving determined which corresponded to different percentages of solid matter in the pulp. The curve in Fig. 3 shows the saving obtained under different conditions, and proves that thickening the pulp beyond a certain limit decreases quite rapidly the economy of a vanner. It should be observed that in these experiments the settling of the pulp has been carried much farther than

would be done in practice. A pulp of this material containing 24 per cent. of solid has almost the consistency of syrup. To convey an idea of the thickness of the feed, the curve, Fig. 4, has been drawn, showing the settling in a cylinder 22.25 in. high. It proves that it takes a long time to settle pulp to this consistency; consequently, if this was carried out in practice it would require a very large settling-capacity.

The adjustments of the vanners in these experiments having been kept the same, the question arises, is it right to assume that the adjustments found the best in the former tests will also be the best for thickened feed? In other words, may not a very

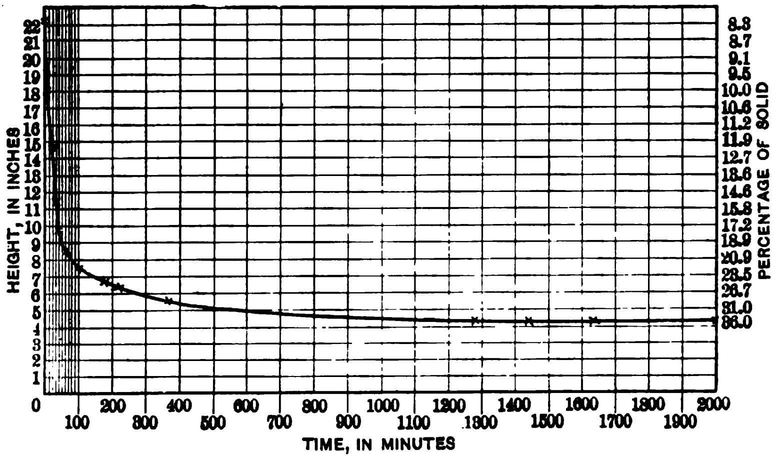
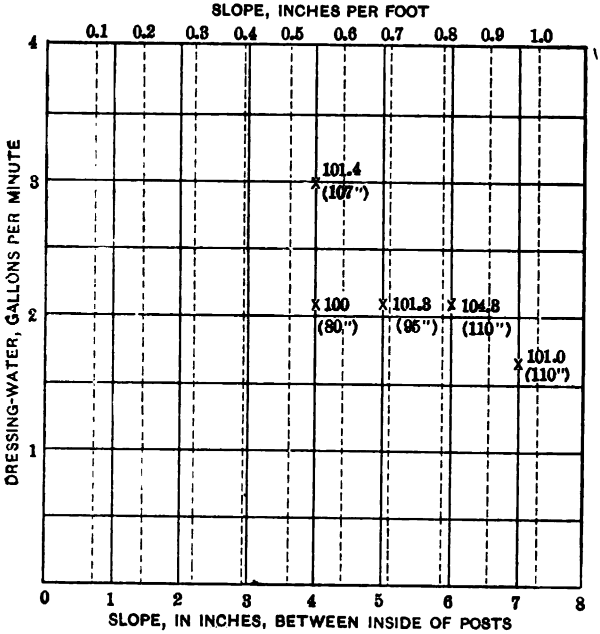


FIG. 4.—SETTLING OF LOWER VANNER FEED (D. C. M. CO. CONCENTRATOR) IN 22.25-IN. CYLINDER.

thick feed require a different adjustment? This question was investigated by the tests represented in Fig. 5, with a pulp averaging 25.5 per cent. of solids. At first, two chances for improvement seemed to exist: one, the use of a higher slope, in order, as far as possible, to spread out the thick pulp and effect a greater contact with the belt; the other, the use of much dressing-water in order to counteract the deficiency of water in the pulp. The outcome of the experiments showed that spreading out the pulp by increasing the slope does not counteract the harmful effects of the deficiency of water in the pulp, but it also showed that an increase of the quantity of dressing-water offsets the disadvantages of a too-thick pulp.

Since the data for the saving in Fig. 5 are expressed by the saving of another vanner with equally thick pulp, the high saving of 107.4 per cent. means only so much more saving than can be made on a vanner running with 4-in. slope (0.544 in. per ft.) and 2.2 gal. of dressing-water treating feed of the same thickness. However, the results show that an extremely thick



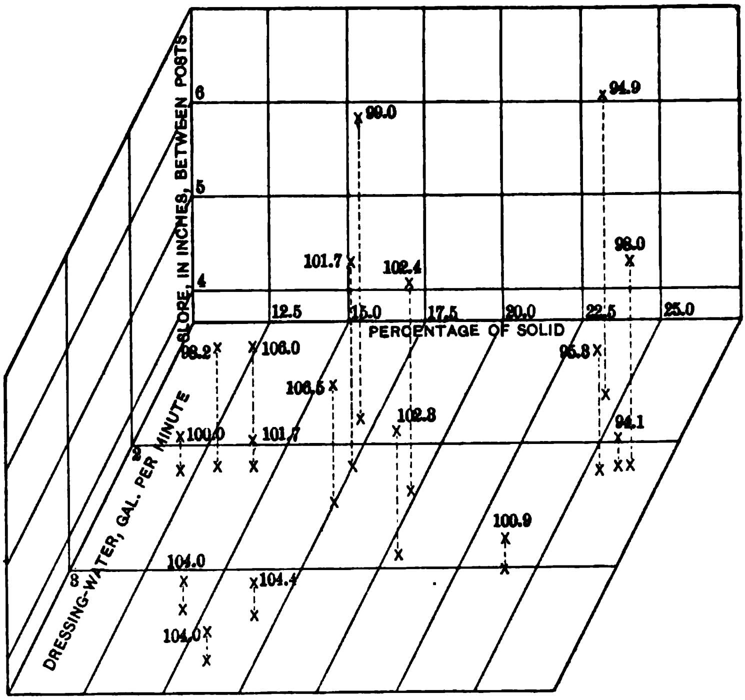
Smooth belt, 210 (1-in.) strokes per minute.
Feed: Lower vanner feed of D. C. M. Co. concentrator.
25.5 per cent. solid (average).
9 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.46 per cent. copper.
6.2 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made with 4-in slope between posts (0.544 in. per foot) and 2.2 gal. dressing-water per minute.
Figures in parentheses mean travel of belt in inches per minute.

FIG. 5.—VARIATION OF SAVING WITH DRESSING-WATER AND SLOPE FOR VERY THICK FEED.

feed can be treated fairly well with a large quantity of dressing-water. The tests, Figs. 4 and 5, are also shown in Fig. 6, together with the results of other test-runs. It seems from Fig. 6 that the best results can be obtained with a slope of 5 in. and a pulp-thickness of about 16 per cent. of solid.

Fig. 6, the saving effected, is represented in its relation to

the three variable quantities which determine it—namely, the percentage of solid in the feed, the slope of belt, and the amount of dressing-water. The saving made by each combination tested is expressed by the figure attached to the point representing this combination. The distance of each of these points from the base-plane, indicated by the length of the per-



Smooth belt, 210 (1-in.) strokes per minute.
Feed : Lower vanner feed of D. C. M. Co. concentrator.
9 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.47 per cent. copper in feed.
7.10 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made with 4-in. slope between posts (0.544 in. per foot) and 2.2 gal. dressing-water per minute.

FIG. 6.—TESTS WITH THICKENED FEED.

pendiculars (the dotted lines), expresses the slope; the position of the foot-point of these perpendiculars in the base-plane shows the corresponding figures for the quantity of dressing-water and for the percentage of solid in the feed.

These results, so far as the percentage of water in the feed is concerned, cannot be applied to other ores, since the consist-

ency best suited for treatment on concentrating-machines depends largely on the composition of the ore (the percentage of clay, etc.). I think, however, that a pulp of the same degree of fineness which will settle with the same velocity as the samples treated would give nearly the same results; also, that results for pulps of equal settling-velocity would yield a better comparison than pulps of equal percentage of solids.

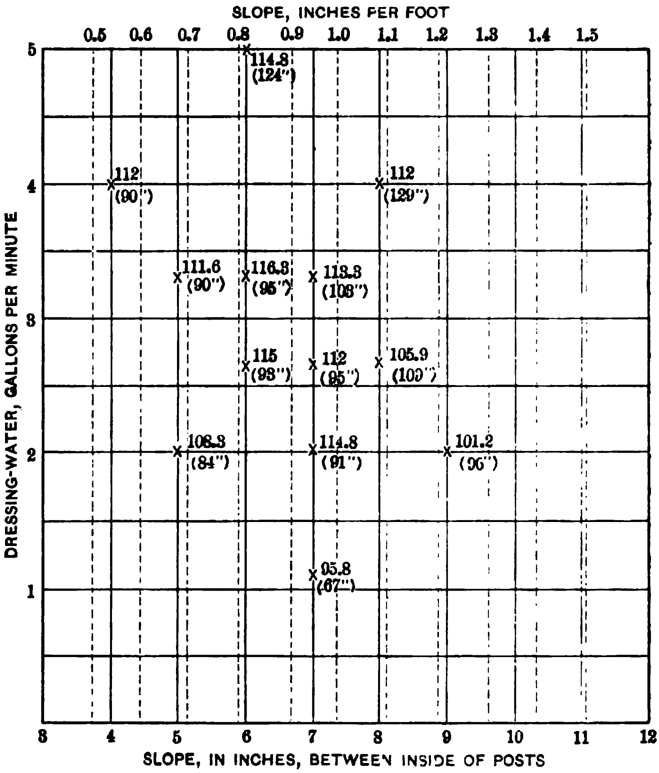
Other tests were made to determine the best adjustments for a corrugated belt treating this slime-feed in the same way in which the smooth belt had been tested before. Since these experiments were carried out before the preceding series, no use was made of the fact that the saving could be improved by slightly thickening the pulp. Otherwise, it might have been more logical to conduct these experiments with the corrugated belt with a somewhat thickened feed. The results of these tests are represented in Fig. 7, in which the saving is expressed in percentages of the saving made on the vanner with the smooth belt, which was taken as the standard. The corrugated belt used was an old one, the corrugations of which were much worn and rounded by long use.

From the results given in Fig. 7, it follows that the best slope for this belt is 6 in. between the posts, or 0.816 in. per ft. The belt-speed which had to be used in each run is added in parentheses to the figures representing the saving. The speed necessary for a good saving is not quite as high as that of the smooth belt, probably on account of the lower number of side-strokes used. The figures show also that a far larger amount of dressing-water was necessary than for the tests with the smooth belt.

All the tests had shown that a high belt-speed is most advantageous in saving slime. Experiments were then tried to ascertain the possibility of increasing the belt-speed, and accordingly the saving, without giving the belt, as a whole, too much fall or applying too much dressing-water, simply by raising the front part only of the belt. The higher fall of the front end was gained by raising the front roller about 1 in. and the second roller enough barely to support the belt. The resulting increase in the rate of belt-speed amounted to about 10 in. per min. In every case, however, the effect on the saving was less than 1 per cent. In one case there was a small gain, in two cases a small loss, so that it seems safe to conclude that the effect of

this change is very small and hardly larger than the errors connected with these tests.

Tests were made to determine the influence of the shaking-motion on the saving made by a vanner. Some attempts had

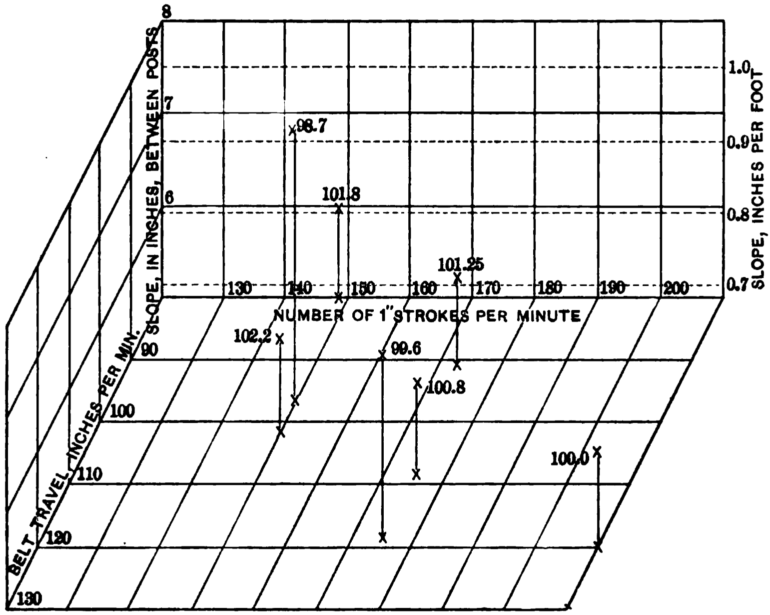


Corrugated belt, 190 (1-in.) strokes per minute.
Feed: Lower vanner feed of D. C. M. Co. concentrator.
12.64 per cent. solid.
9.0 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.4 per cent. copper.
8.43 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made on standard vanner with smooth belt, 229 (1-in.) strokes per minute, 5-in. slope between posts, 123 in. belt-travel per minute.
Figures in parentheses mean travel of belt in inches per minute.

FIG. 7.—VARIATION OF SAVING WITH DRESSING-WATER AND SLOPE.

been made before to gain information on this question. But at that time the power for the experimental machines was quite variable, and the only result obtained was that any considerable reduction in the shaking-motion was practically impossible,

because the machines were influenced too much by the changes in the power. Later, better power was secured, and although variations of 2 per cent. to either side in the number of strokes still occurred frequently, it seemed to be improvement enough to consider a reduction in the speed of the shaking-motion. The results, Fig. 8, do not show a great improvement to be



Corrugated belt (D. C. form).
Feed: Lower vanner feed of D. C. M. Co. concentrator.
14.4 per cent. solid.
10.0 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.44 per cent. copper.
8.80 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made on vanner with corrugated belt (D. C. form), 210 (1-in.) strokes per minute, 6-in. slope between posts (0.816 in. per foot), 100 in. belt-travel per minute.

FIG. 8.—VARIATION OF SAVING WITH NUMBER OF STROKES.

gained by reducing the speed, but indicate that a reduction of the number of side-strokes to about 150, with a considerable increase in the amount of dressing-water in order to gain high belt-speed, would be an improvement. A reduced side-motion requires more dressing-water to produce the same belt-speed.

Very probably such an extensive reduction would prove impracticable with our power-conditions, as one man has to operate

a large number of machines; but doubtless some reduction would be beneficial, especially with our high rate of shaking-motion, which, in spite of continuous attention, shakes the machines loose very soon, and this, of course, means loss.

As already mentioned, Fig. 7 shows the saving made by the corrugated belt in percentages of the saving made by the smooth belt. These runs were originally not intended to determine which belt made the best saving, but only to find the best setting for the corrugated belt, using the smooth belt as a standard in all these tests. The saving made on the corrugated belt was, however, in all these tests so much higher that it seemed safe to conclude that the corrugated belt was better than the smooth belt for the treatment of slime-feed. This result was doubted by many experienced in the concentration of copper-ores, and a second contest was arranged in which I operated the corrugated belt against several experienced concentrating-men using a smooth belt. The smooth belt was nearly new, having served in the concentrator for some months. The corrugated belt was the one used in the former tests, and before this had been in constant use for a long time in another mill, so that it was not in first-class condition. In some places the riffles had been worn away almost completely, while in others they still stood out prominently. Besides, there were numerous bad places, due to rough handling in shipping and repeated putting on and taking off the belt. At the bottom of the riffles the canvas was exposed.

In the first test of the new series the slope of the smooth belt was the same as before, 2 in. (0.272 in. per ft.). The results obtained were:

Lower Vanner Feed.—10 per cent. on 200-mesh screen; 12.31 per cent. of solid matter; 1.30 per cent. of copper; 8.5 tons (dry) per 24 hr. Running time, 6 hr. 40 min.

	Smooth Belt.	Corrugated Belt.
Slope between posts, inches,	2	6
Slope per foot, inches,	0.272	0.816
Belt-travel per minute, inches,	60	120
Strokes per minute,	234	194
Concentrates, dry weight, pounds,	82.63	160.90
Concentrates, Cu, per cent.,	20.66	17.39
Copper saved, pounds,	17.07	27.98
Tailings-assay, Cu, per cent.,	0.97	0.79

Saving of corrugated belt in percentage of smooth belt, 163.9 per cent.

In the following tests no limitations were set to the adjustments; 12 days were devoted to these tests, including work with the table mentioned before, which received feed from the same feed-distributer. The average results were:

Lower Vanner Feed.—8.81 per cent. on 200-mesh screen; 11.91 per cent. of solid matter; 1.49 per cent. copper; 8.99 tons (dry) per 24 hr.; running-time, 83 hr. 30 min.

	Smooth Belt.	Corrugated Belt.
Slope between posts, inches,	3.96	6.00
Slope per foot, inches,	0.538	0.816
Belt-travel per minute, inches,	100	116
Strokes per minute,	204	195
Concentrates, dry weight, pounds,	1,438.6	1,622.2
Copper saved, pounds,	266.23	300.35
Concentrates, Cu, per cent.,	18.51	18.51
Tailings-assay, Cu, per cent.,	0.824	0.797

Saving of corrugated belt in percentage of smooth belt, 112.8.

For a further comparison between the work of the corrugated and the smooth belt, a test was made using as feed the tailings from the slime-vanners (smooth belts) in the concentrator. The results of this test were:

Lower Vanner Feed.—Tailings from lower vanners, running with smooth belts, about 3.5 in. slope between posts (0.476 in. per ft.); 9 per cent. on 200-mesh screen; 10.73 per cent. of solid; 0.77 per cent. of copper; 6.90 tons per 24 hr.; running-time, 10 hr. 30 min.

	Smooth Belt.	Corrugated Belt.
Slope between posts, inches,	4	6
Slope per foot, inches,	0.544	0.816
Belt-travel per minute, inches,	95	98
Strokes per minute,	207	207
Concentrates, dry weight, pounds,	54.5	82.56
Concentrates, Cu, per cent.,	12.09	10.46
Copper saved, pounds,	6.59	8.64
Tailings-assay, Cu, per cent.,	0.68	0.67

Saving of corrugated belt in percentage of smooth belt, 131.7.

This result confirmed the experience gained in the first tests, that the saving effected by the corrugated belt on these slime-tailings far exceeds the saving made by the smooth belt.

The tests with the corrugated belt had been made using the old belt, the corrugations of which were greatly worn down. Since the results were so very encouraging, a new belt of the same pattern and having sharp corrugations was secured and tested. The saving effected was extremely disappointing. It

was not possible to get even as good results as with a smooth belt. Later, a large number of belts were made with exactly the same shape of riffles as had been formed by the wear.

With one of these belts tests were run against both the smooth and the old corrugated belt, which showed that the new belt, after it had been on the machines for a couple of weeks, was just as good a saver of mineral as the old one, and was considerably superior to the smooth belt. One difference existed—namely, that the old belt, probably on account of its roughness and of the exposure of the canvas at the bottom of the riffles, carried some very fine slimes into the concentrate, which, however, was of rather low grade, and did not influence the saving very much.

For a test on a larger scale, 10 machines, equipped with the new corrugated belts, were run against 10 machines with smooth belts. The concentrate from each group of machines was caught in a wooden tank large enough to hold all the concentrate produced in the course of a week. Five tests were made, the first one lasting seven days, the others five days each. The results of the first test (which favored the corrugated belt) were not accepted, since there was some doubt of the correctness of the work. In the fourth and fifth tests, the feed-pipes which had been supplying the feed for the smooth belts in the first and second tests supplied the corrugated belts, and *vice versâ*, in order to eliminate possible inaccuracies of the distributer. The smooth belts used for the tests had been running in the concentrator for some time. The rubber was worn off from the back surface of some of the belts and the canvas exposed. To prevent the removal of mineral from the vanner-box on this rough surface, spray-water was used to keep the surface clean.

The average result of these tests, occupying 20 days altogether, was that the corrugated belts produced 123.2 per cent. of the copper produced on the smooth belts. The grade of concentrate on the corrugated belts was 20.04 per cent. of copper, as compared with 19.39 per cent. on the smooth belts.

The corrugated belts were set with 4.92 in. slope (0.669 in. per ft.) and the smooth belts with 3.56 in. (0.484 in. per ft.), which is a little less than the slopes found best in the experiments. But the ordinary load in the mill is smaller than the

load used on the experimental machines, so that a gentler slope may be justified. The results obtained are decisive enough to permit the statement that the corrugated belt of the form described above has proved a better slime-saver than a smooth belt.

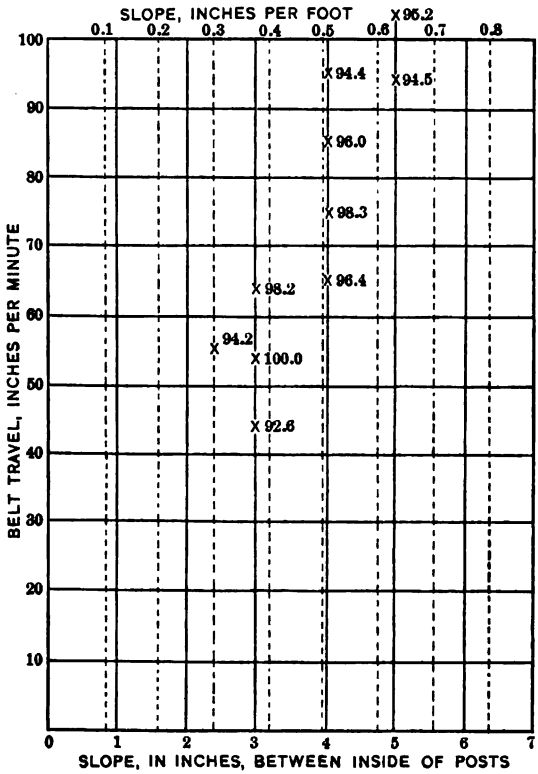
It is often pointed out as an objection to a corrugated belt that it will make a dirtier concentrate, although it may make a better saving. Expressed so broadly, this statement is certainly incorrect. Since sand cannot be noticed as easily as on a smooth belt, it requires practice to produce a uniformly clean concentrate on a corrugated belt. But there is not the least doubt that, if the concentrate made on a given machine is not clean enough, it can be raised to any degree of purity by the ordinary adjustments of vanners. My tests show that the new corrugated belt produces a larger amount of concentrate of the same grade than does a smooth belt.

Calculations based on the result of the test determining the saving that could be made by re-treating the tailings from the slime-vanners on corrugated belts had shown that it would hardly pay to install additional machines for this purpose with copper at a price of $0.13 per lb. Replacing the smooth belts in the concentrator by corrugated belts will improve the saving of the vanners and make the re-treatment of tailings from the slime-vanners decidedly uneconomical.

Possibly it may be profitable to classify the tails thoroughly before attempting re-treatment, but since these slimes contain only a small percentage of material that will stay on a 200-mesh screen, it is not probable that this suggestion will lead to any improvement. Besides, if classification yields a better saving, it would be better to provide a thorough classification for the vanner-feed.

Another way of raising the saving of the slime-vanners is to lower the tonnage treated per machine, and with this end in view experiments were undertaken to establish the relation between the saving made and the load carried on these vanners. The feed-distributer was provided for these tests with compartments of unequal size, so that more feed could be sent to one vanner than to the other one. The proportion in which the feed was distributed was carefully determined. The results of these tests are represented graphically in Fig. 10, which shows

that within the limits tested the vanner carrying the lighter load makes the better saving. The improvement resulting from reducing the load is not very great. For the present grade of milling ore, with copper at 12.5 cents, and with the present cost



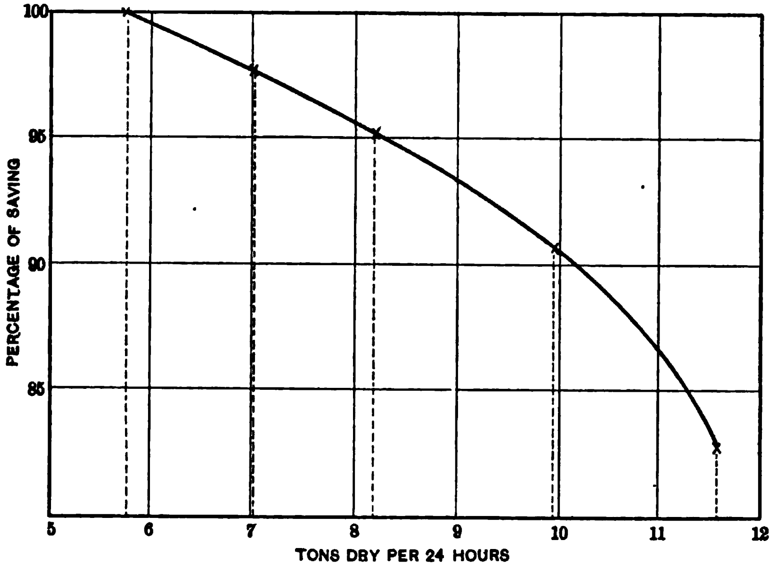
Corrugated belt, 210 (1-in.) strokes per minute.
Feed: Upper vanner feed East, D. C. M. Co. concentrator.
12.65 per cent. solid.
27.2 per cent. on 100-mesh screen, 43.6 per cent. (cumulative) on 200-mesh screen (Denver Fire Clay Co.).
1.51 per cent. copper.
6.74 tons dry per 24 hours.
Saving is expressed in percentage of saving made on corrugated-belt vanner with 210 (1-in.) strokes per minute, 3-in. slope between posts (0.408 in. per foot), 54 in. belt-travel per minute.

FIG. 9.—TESTS WITH COARSER FEED.

of labor, power, repairs, etc., the most economical load for the new corrugated belts is between 8 and 9 tons, which shows that under existing conditions a reduction of the present load is not advisable. For higher copper-prices, however, a smaller load

would be more economical. At 20 cents the greatest economy is obtained with a load of about 7 tons.

The slope which is to be given to corrugated belts depends very largely on the kind of material treated. Formerly it was the rule at the Detroit concentrator, and I suppose many others, to treat the finest feed with the lowest, and the coarsest feed with the highest, slope. That the contrary is rational has been



Corrugated belt (D. C. form), 210 (1-in.) strokes per minute, 6-in. slope between posts (0.816 in. per foot), 120 in. belt-travel per minute.
Feed: Lower vanner feed of D. C. M. Co. concentrator.
14.7 per cent. solid.
10.0 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.31 per cent. copper.
Saving is expressed in percentage of saving made on a vanner with the same adjustments carrying a load of 5.77 tons dry per 24 hours.

FIG. 10.—VARIATION OF SAVING WITH LOAD.

held by Richards and others. The experiments, graphically represented by Fig. 9, show that on a coarse feed the higher slope gives poorer savings than a lower slope. The fact that a feed not very different requires such a different treatment, seems to point to close classification as an improvement for vanner-work, and to proper adjustments of each machine to the pulp treated.

This question is still undecided, but I hope that some light

has been thrown on a few points by the work described in this paper. If nothing else, it shows that it pays to study questions of this character.

NOTE (July, 1909).—Since writing this paper some additional tests have been made, in which the corrugated belts showed no marked superiority over smooth belts. These tests were made under mill conditions, 10 corrugated-belt machines running against 10 others with selected smooth belts, both sets of machines operated with high belt-speed. The feed was somewhat coarser than in the tests referred to in the paper. Although, therefore, the results are not exactly comparable, they could be interpreted as throwing some doubt on the correctness of the previous tests investigating this point. For this reason I have intended for a long time to repeat these tests. But as some changes in the operation of the mill have made it impossible so far to obtain the same kind of feed as used in the former tests, and as there may not be an opportunity of carrying out this plan for some time, I thought it better not to delay the publication any longer. I felt, however, that I should call attention to this apparent discrepancy.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Metal-Losses in Copper-Slags.

BY LEWIS T. WRIGHT, SAN FRANCISCO, CAL.

(New Haven Meeting, February, 1909.)

IT is commonly believed by metallurgists that in copper-smelting, the copper in the slags, which is irreducible by continued smelting, is retained in the form of "prills" of matte.

I have frequently held well-settled slag in a molten condition for a long period without being able thereby to reduce the copper-content. The slag acted as though it contained a minimum of dissolved or combined copper that could not be settled out by gravity. I have used reagents, but without satisfying myself in what form this copper existed. The same slags, by fine grinding and elutriation, could not be separated into portions containing more or less copper than the average content.

By treating copper-slags in an electric furnace an impure button of copper was produced, but the high temperature and the strong reducing-action of the furnace were influences that suggest an explanation of a result not obtainable with ordinary smelting-temperatures.

If all the copper in the slag were in the form of copper-matte, and not existing as a dissolved compound with some of the elements of the slag, then the other metals in the matte, such as gold and silver, should occur in the slag in the same ratio to the copper as to the copper in the accompanying matte.

This reasoning led me to study the ratio of metals in the products of smelting, and, apart from the issues discussed in this paper, the research has proved both interesting and illuminative.

On the assumption of a similar ratio of metals in matte and slag, if the percentage of copper in the slag accompanying a matte containing 50 oz. of silver and 1 oz. of gold per ton of copper was 0.3 per cent., or 96 oz. per ton of slag, there should be found 0.15 oz. of silver and 0.003 oz. of gold per ton of slag; but, in my experience, there is less silver and much

less gold in the slag than is required by the assumed law of similar metal-ratios.

In order to investigate this point, many accurate assays of matte and slag, covering long periods of time, were grouped into series so as to obtain an average effect of contemporaneity of occurrence of both products, viz., matte and slag, and it was found that the greater the concentration of gold and silver in the copper-matte, the less, relatively, is the copper-gold-silver ratio in the slag.

Some of the results obtained, and which are now given in Table I., illustrate the general nature of the ratios thus discovered.

TABLE I.—*Gold- and Silver-Content of Copper-Matte and Accompanying Slag.*

Troy ounces of gold and silver per ton of copper.

Matte.		Slag.		Ratios.	
Gold.	Silver.	Gold.	Silver.	Gold in Slag. / Gold in Matte.	Silver in Slag. / Silver in Matte.
Ounces.	Ounces.	Ounces.	Ounces.		
27.9	62.8	8.65	48.9	0.31	0.78
3.5	33.2	2.3	30.9	0.66	0.93
2.5	166.4	1.62	127.8	0.65	0.76
3.04	152.0	2.10	116.0	0.69	0.76

It should be noted that in the above table the precious metals are not stated in the ordinary manner in ounces per ton of matte and slag respectively, but in ounces per ton of copper. Thus a 50-per cent. matte containing 13.95 oz. of gold per ton of matte is shown in the table as containing 27.9 oz. of gold. In the same manner, the slag stated in the table as containing 8.65 oz. of gold would, if it contained 0.3 per cent. of copper, in the ordinary manner be stated as containing 0.02595 oz. of gold per ton of slag.

If all the copper in the slag existed as matte entrained in the slag, the metal-ratio should be the same as in the matte; but this is not so in practice.

David Browne, of Sudbury, Ontario, informs me that the nickel-copper ratio of matte is not the same as that of the accompanying slag.

Although there is not enough data as to the form, or forms,

in which the irreducible copper exists in the slag to justify more than a suspension of judgment on this point, still the relations noted indicate that the metals are dissolved in the molten slag.

The reversible chemical reaction, $Cu_2S \leftrightarrows CuS + Cu$, explains the presence of free copper in solid copper-mattes, which is more marked in high-grade than in low-grade mattes. If free copper, also, exists in the molten matte and in increased quantity with the higher grade, the general observation that the copper carried away in well-settled slag increased with the

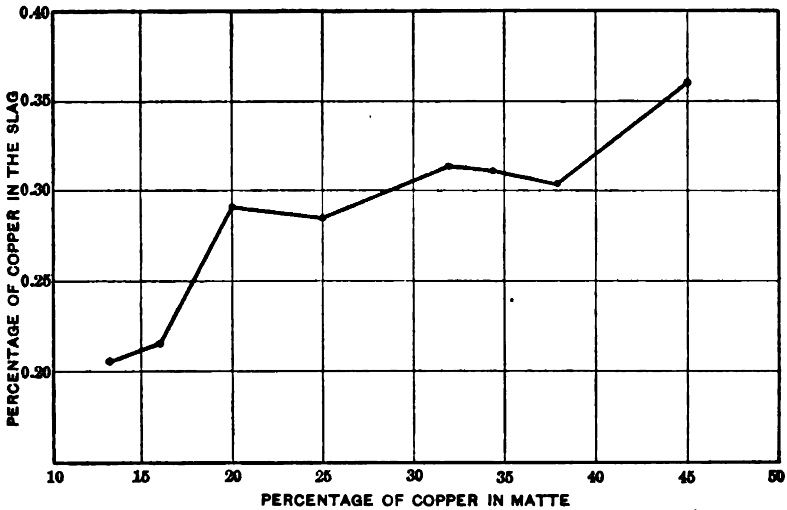


FIG. 1.—CURVE SHOWING VARIATION OF COPPER IN COPPER-MATTES AND ACCOMPANYING SLAGS.

grade of the accompanying matte is more suggestive of the influence of some physico-chemical property than of the fortuities of mechanical separation. Between the limits of 15- and 45-per cent. copper-matte the accompanying slags increase by an average of 0.005 per cent. of copper per 1 per cent. of increase of copper in the matte.

This increase, given in Fig. 1 as a curve, shows a well-marked flattening between 20- and 37-per cent. copper-matte, and depressions at 25- and 37-per cent. matte.

The distribution of a soluble body in two solvents is independent both of the relative amounts of the two solvents present and of the absolute concentration. The substance is dis-

tributed so that the ratio of its concentration in each solvent is constant. If this so-called "law of distribution" were strictly applicable to copper-mattes and accompanying slags, and there were no interfering or modifying influences, then the ratios given in Table I. would be nearer to 1 than they are.

Other things being equal, a difference of 1 per cent. of SiO_2 in the slag makes a regular difference of about 0.01 per cent. in its copper-content. The regularity of this occurrence is more suggestive of solution than of differences due to more or less perfect settling.

If slag formed in making a matte having a certain metal-ratio be kept in molten contact with matte possessing a different metal-ratio, the slag will acquire a new metal-ratio due to that of the latter matte. The matte and slag appear to act as solvents and divide the metals accordingly.

If slag formed in making matte carrying a large quantity of precious metals be kept in molten contact with matte containing a small quantity of precious metals, the silver and gold will go out of the slag into the matte, although the percentage of copper in the slag, other things being equal, will remain constant. In this way the precious metals may be almost entirely removed from copper-slags.

I applied this knowledge of metal-ratios to great advantage in smelting copper-ores.

In a series of three reverberatory furnaces the middle one was built with a hearth lower than the others. All the slag from the outside furnaces is tapped through the middle furnace. The charge smelted in the outside furnaces yields a matte having high precious-metal values, while that in the middle and lower furnace yields a matte low in precious-metal values. In this way all the slag produced in either the outer or the inner furnaces can be discharged low in copper, gold, and silver.

The middle, or "cleaning," furnace smelted 6 per cent. more ore-charge than the two outer furnaces and consumed 17 per cent. less fuel, giving a total saving of 21.7 per cent. of fuel. The slag from the outer furnaces was not considered a part of the "ore-charge."

By using this method of smelting copper-ores the precious metals are recovered with an increased furnace-output and a large economy in fuel-consumption.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Development in the Size and Shape of Blast-Furnaces in the Lehigh Valley, as Shown by the Furnaces at the Glendon Iron Works.

BY FRANK FIRMSTONE, EASTON, PA.

(New Haven Meeting, February, 1909.)

In the summer of 1842 my father, William Firmstone, was engaged by Charles Jackson, Jr., of Boston, to examine the conditions in the Lehigh valley as a site for blast-furnaces using anthracite for fuel. In consequence of his report, he was further engaged by Mr. Jackson to build a furnace for him and his partners on the Lehigh canal, 2 miles above the mouth of the river at Easton. Work was begun in the fall of 1842, and the first furnace blown-in in 1844. The history of the works, therefore, from 1844 to 1887, when my own connection with them ceased, covers only four years less than the whole period of the rise, culmination, and commencement of the decline in the smelting of iron with anthracite in America, the beginning of which, in a commercial sense, may be put in 1839.[1]

Although this paper, as the title indicates, is concerned almost exclusively with the furnaces at Glendon, still what was done there was more or less influenced by current opinion in the district, and even reflects, to some extent, the changes in opinion and practice with mineral fuel the world over.

No. 1 Furnace was built of red brick, on four piers, had three tuyeres and fore-hearth and tymp, as was universal with furnaces using mineral fuel until the introduction of Lürmann's cinder-tuyere. The dimensions and profile are shown in Figs. 1 and 2. The first hearth (Fig. 1) was of sandstone; all after were of fire-brick (Fig. 2). The profile was no doubt derived directly from Gibbons's furnace,[2] for it appears from W. Firmstone's note-books that in October, 1840, he visited "Gibbons' new furnaces building at Corbyn Hall," and I have an old drawing,

[1] *Trans.*, iii., 153 (1874–75).
[2] Percy, *Metallurgy of Iron and Steel*, p. 479 (1864).

made by him, marked "Corbyn Hall, 1841," practically the same as the figure above named in Percy. Although I have called this, with others, the Gibbons profile, essentially the same shape had been used in Sweden long before,[3] and something not very different in England at a still earlier date.[4]

Although what had already been done in this country, especially by David Thomas at the Crane works at Catasauqua, would have warranted building a much larger furnace, Mr. Jackson desired to make the first trial on a very modest scale. No. 1 worked well, and the building of No. 2 was begun in 1844 and the furnace blown-in in 1845. This furnace worked very well, used less fuel per ton of iron than No. 1, and naturally made much more iron. On the whole, it was for many years the best furnace at Glendon, especially in the very important point of regularity. The profile, Fig. 3, approximates the Gibbons shape, but the proportions are somewhat different from No. 1 and from Corbyn Hall. The dimensions were: Corbyn Hall, $H/d = 3.56$; No. 1, $H/d = 3.63$; No. 2, $H/d = 3.21$. No. 2 had four tuyeres. Excepting an increase in the diameter at the tuyeres from 4 ft. 9 in. to 5 ft., and a consequent slight steepening of the boshes, no important change was made in the profile of this furnace until 1869. Both No. 1 and No. 2 were blown by water-power, and the hot-blast ovens were fired with anthracite, as was the case at Catasauqua.[5] None of the gas was then utilized at Glendon.

The works were further increased in 1849 and 1850 by tearing down No. 1 and building a larger furnace with five tuyeres on the same site, and by building a third furnace, also having five tuyeres. By this time the use of the waste gases was well understood, and, accordingly, boilers and steam blowing-engines were erected to supply No. 3 Furnace, and to give more blast to Nos. 1 and 2, and all the hot-blast ovens were altered to be heated by the gases. Both boilers and hot-blast ovens were raised, some on cast-iron columns and some on masonry piers, to nearly the level of the flues, by means of which the portion of gas used was taken from the furnaces, very much as shown in

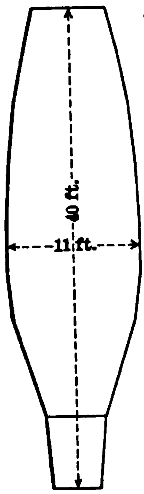
[3] Braune, *Jern-Kontorets Annaler*, vol. lix., pp. 34, 35 (1904); and Jars, quoted in Beck's *Geschichte des Eisens*, part 3, pp. 355, 356.

[4] Beck, *ibid.*, part 2, p. 970.

[5] S. Thomas, *Trans.*, xxix., 908 (1899).

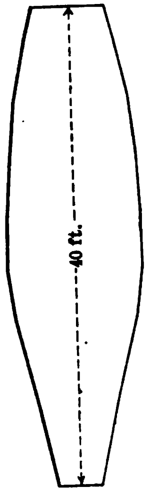
Mr. Thomas's paper, above cited, and which was, in fact, the standard construction for anthracite-furnaces in eastern Pennsylvania until the general introduction of closed tops in 1869 and 1870.

In the late '30's and early '40's there was, apparently, a strong tendency to increase the volume of the furnace, particularly ot the upper part, and Gibbons had proved this to be correct



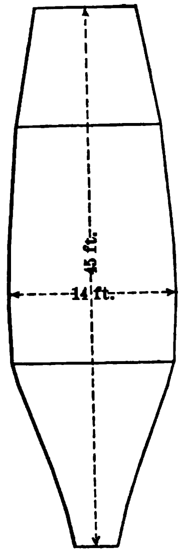
Blown-in, Mar. 15, 1844.
Blown-out, July 17, 1844.
Iron per week, tons, 52.30.
Average grade of iron, 1.68.
Fuel per ton pig, tons, 2.28.
Blast-temp., 500°-600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 3.

FIG. 1.—FURNACE No. 1.



Blown-in, Sept. 5, 1844.
Blown-out, May 16, 1846.
Iron per week, tons, 57.46.
Average grade of iron, 1.89.
Fuel per ton pig, tons, 2.04.
Blast-temp., 500°-600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 3.

FIG. 2.—FURNACE No. 1.



Blown-in, May 25, 1845.
Blown-out, Aug. 24, 1847.
Iron per week, tons, 68.36.
Average grade of iron, 3.27.
Fuel per ton pig, tons, 1.95.
Blast-temp., 500°-600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 4.

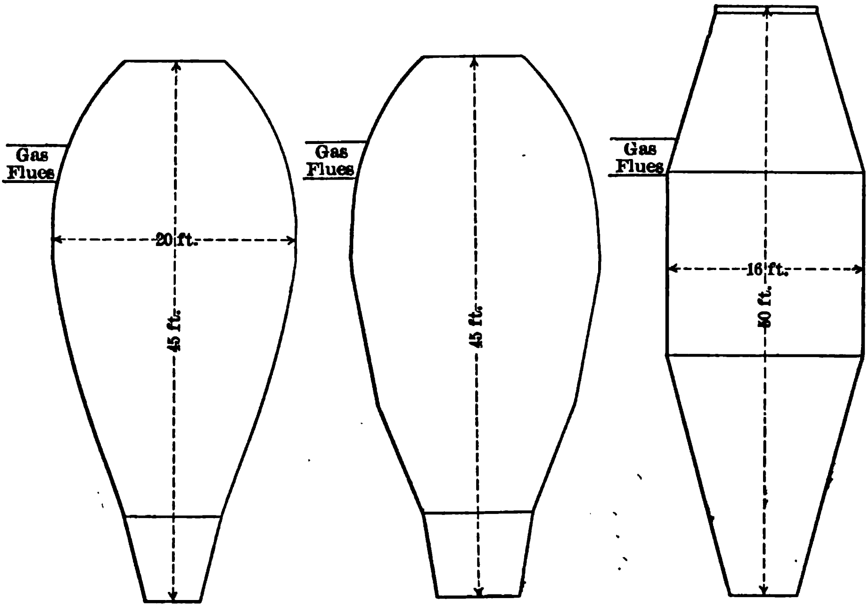
FIG. 3.—FURNACE No. 2.

within limits by actual trial.[6] The same tendency was probably responsible for making furnaces cylindrical at the widest part for considerable heights, as, for instance, the furnace at the Crane works, shown in Mr. Thomas's paper.[7] In this connection, the following extract from a private note-book of W. Firmstone is of interest:

"Liverpool, Sept. 10, 1840, 3:30 p.m., went on change. Saw Mr. Jeavons. Said his furnace in South Wales had blown in with anthracite coal about a week

[6] Percy, *Metallurgy of Iron and Steel*, p. 477, *seq.* (1864).
[7] *Trans.*, xxix., 909 to 917 (1899).

since, doing well. Said they were a month after firing before they got the burden down to blow. Was surprised when I told him that in the States we blew-in a few hours after firing. Said his furnace was 46 ft. high, 12-ft. boshes; 8-ft. tunnel-head; cylindrical for 15 ft.—Dec. 10, 1840. (Carnbrae.) The furnaces are 8 ft. at top; four filling-places, 12 to 15 at boshes, being pretty straight to near the top, then cupped in all at once. This is the common plan in Scotland."



Blown-in, Mar. 2, 1851.
Blown-out, July 24, 1853.
Iron per week, tons, 114.04.
Average grade of iron, 3.42.
Fuel per ton pig, tons, 1.82.
Blast-temp., 500°–600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 5.

Blown-in, Sept. 20, 1853.
Blown-out, Mar. 28, 1855.
Iron per week, tons, 140.69.
Average grade of iron, 4.14.
Fuel per ton pig, tons, 2.08.
Blast-temp., 500°–600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 5.

Blown-in, July 1, 1855.
Blown-out, July 1, 1857.
Iron per week, tons, 131.47.
Average grade of iron, 2.72.
Fuel per ton pig, tons, 1.91.
Blast-temp., 500°–600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 5.

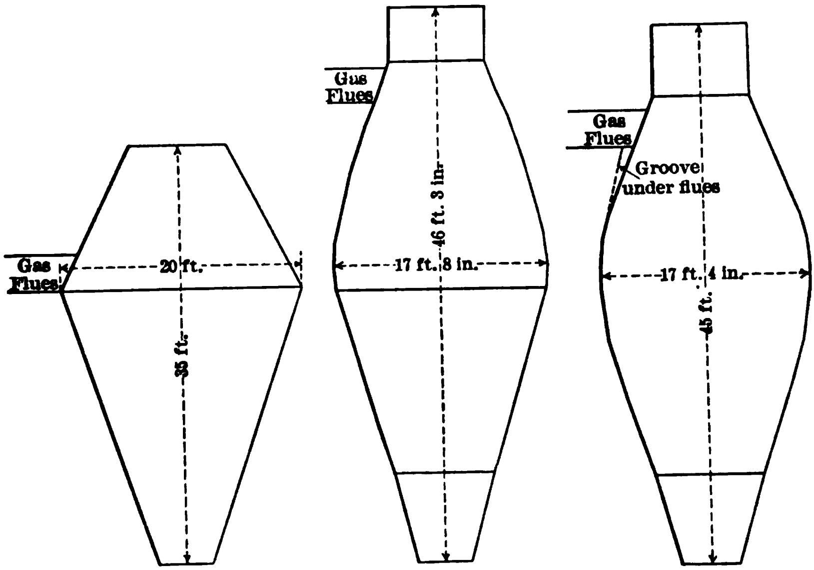
FIGS. 4, 5, AND 6.—FURNACE NO. 1.

This tendency was carried to the extreme in rebuilding No. 1, as shown by Figs. 4 and 5; but the results were not good, and after two blasts the furnace was raised from 45 ft. to 50 ft., and the diameter reduced to 16 ft., Fig. 6, with decided advantage.

There was also a notion at that time, in South Wales at least, that raw-coal- and anthracite-furnaces should be low and of relatively great diameter, no doubt with the expectation that the necessary blast-pressure would be less than with higher furnaces of the same cubic capacity. Such a furnace is shown

in Percy.[8] This notion was tested in building No. 3, Fig. 7, but the results were so decisively against it that it was soon blown-out and altered to the shape shown in Fig. 8.

The ill-effects which attend the dumping of the material in the center of an open-topped furnace from a car with a trap-door bottom, or from a hopper closed by a small cone which is



Blown-in, Oct. 20, 1850.	Blown-in, June 10, 1851.	Blown-in, Aug. 26, 1862.
Blown-out, Mar. 31, 1851.	Blown-out, Aug. 7, 1858.	Blown-out, Apr. 21, 1865.
Iron per week, tons, 103.96.	Iron per week, tons, 109.12.	Iron per week, tons, 154.69.
Average grade of iron, 3.34.	Average grade of iron, 3.97.	Average grade of iron, 4.19.
Fuel per ton pig, tons, 2.30.	Fuel per ton pig, tons, 2.	Fuel per ton pig, tons, 1.82.
Blast-temp., 500°–600°.	Blast-temp., 500°–600°.	Blast-temp., 500°–600°.
Blast-pressure, pounds, 4.	Blast-pressure, pounds, 3.	Blast-pressure, pounds, 4.5.
Number of tuyeres, 5.	Number of tuyeres, 5.	Number of tuyeres, 5.

FIGS. 7, 8, AND 9.—FURNACE NO. 3.

raised to discharge the materials into a closed-topped furnace, have often been noticed, and clearly traced to the rolling of the coarser part of the charge, especially the larger pieces of fuel, to the walls.[9] It is plain that filling with barrows into a furnace like Fig. 4 or Fig. 7 is not quite the same as dumping exactly in the center of the top, yet it is fairly evident that the great and sudden sidewise movement of the materials, in such cases, will result in an accumulation of the larger pieces at the

[8] *Ibid.*, Fig. 103, p. 562 (1864).

[9] De Vathaire, *Études sur les Hauts-Fourneaux*, p. 102 (1866); Chas. Cochrane, *Proceedings of the Institution of Mechanical Engineers*, pp. 163, 164 (1864).

walls, with the attendant bad effects. This unavoidable defect is one reason, probably the principal one, for the failure of such profiles. So far as I know, they are nowhere in use to-day. I have no doubt that for good work a furnace should widen downwards from the top at a moderate rate until about the middle of the total height is reached, but special reasons caused the adoption at that time ('50's and early '60's) in eastern Pennsylvania of shapes widening at what now seems an excessive rate. At all furnaces in the Lehigh valley and, so far as I know, elsewhere in the anthracite-regions, the gas for hot-blast ovens and boilers was then taken from the furnaces by a series of horizontal flues piercing the lining at depths of from 10 to 17 ft. below the open tops of the furnaces. By the rolling of pieces of the materials into the mouths of these flues, and by the accumulation of dust in the interstices of such pieces, the flues were greatly obstructed, and unless they were cleaned out at frequent intervals, a very troublesome operation, the flow of gas was greatly diminished or cut off completely. It is plain that this difficulty will be greatly diminished if the inner mouths of the flues come in a place where the furnace-walls batter inwards towards the top pretty sharply, and partly at least for this reason, many furnaces built at that time were thus drawn in at the top. Furnaces having this shape were not among those, so far as my own knowledge goes, which did notably good work in respect of fuel-consumption and regularity, and the application to them of the cup-and-cone, with no change in profile, no doubt aggravated the difficulties which at first, in most cases, attended the use of closed tops on the Lehigh.

A much better plan than the excessive contraction at the top was adopted at the Crane works, as early as 1850 according to information given me by S. Thomas shortly before his death. It was also used at the Thomas works, which were built in 1854–55. The furnaces widened from the top downwards at a moderate rate, but grooves, the width of the flues, were formed under each flue, so that a line following the natural slope of the materials drawn from the top of the flue did not cut the bottom but came into the groove; thus no solid matter but the flue-dust could enter them, and they were easily cleaned. The excessive sidewise rolling of the materials was confined to

the space in front of each flue instead of occurring all around the circumference. This construction is clearly shown in the drawings of the Thomas works in Percy;[10] also Wedding.[11] The same plan was used at the raw-coal furnace at Canal Dover, Ohio, by David Thomas, Jr., and was employed at some of the raw-coal furnaces in the Shenango and Mahoning valleys in the late '60's and early '70's, no doubt derived from Canal Dover.

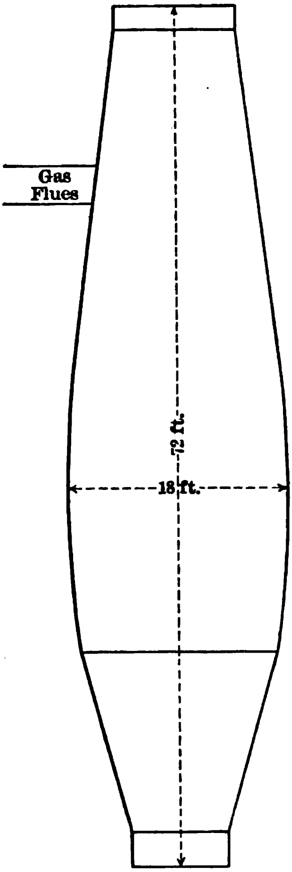
No. 3 Furnace was altered to this plan in 1862, but, as Fig. 9 shows, the top was drawn in excessively in order to get sufficiently deep grooves, and, in fact, although the furnace thus altered was an excellent gas-producer and greatly helped the other furnaces in this way, yet the consumption of fuel was always larger than in Nos. 1 and 2, in spite of a greater proportion of mottled- and white-iron in the product; the working also was decidedly less regular.

No important changes were made at Glendon from the '50's until 1867 and 1868, but at the Thomas works at Hokendauqua, which were built in 1854–55, the furnaces were made 60 ft. high. These furnaces are fully represented in the plates in Percy and Wedding above cited. Very full statistics concerning them were published by Prof. John A. Church in 1875.[12] The figures for fuel-consumption there given are not directly comparable with those here given for Glendon for like dates, because the furnaces at Glendon were burdened to make as much gray-forge as possible, while the object at Hokendauqua was to produce a very soft gray foundry-iron. Moreover, the ore-mixture at Glendon, consisting of more than half magnetic ores, was richer than that at Hokendauqua, which contained more than half of rather lean brown ores.

No. 5 Furnace at Glendon was designed and built in 1867-68. By that time the great saving in fuel which had resulted from increasing the cubic content, and especially from greater height in the furnaces, in the Cleveland district, had attracted general attention. The first design for No. 5 was for a furnace 60 ft. high and 18 ft. in diameter, but this was changed during construction to 72 ft. high and 18 ft. in diameter, the profile being

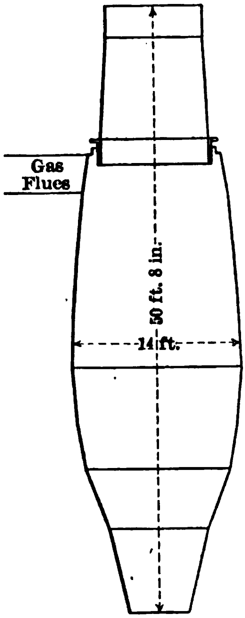
[10] *Ibid.*, p. 380, and the folding plates at end of volume (1864).
[11] *Ausführliches Handbuch der Eisenhüttenkunde*, 2d ed., vol. iii., p. 17 (1906).
[12] *Trans.*, iv., 223 (1875–76).

as in Fig. 10. This profile differs but little from the Gibbons shape used in No. 1 in 1842, but H/d = 4, and the furnace belongs to Gruner's *élancé* type.



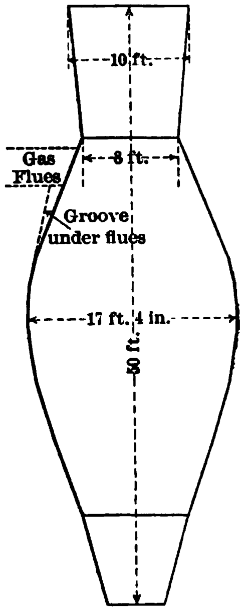
Blown-in, Jan. 7, 1869.
Blown-out, Feb. 28, 1871.
Iron per week, tons, 257.00.
Average grade of iron, 3.72.
Fuel per ton pig, tons, 1.35.
Blast-temp., 500°–650°.
Blast-pressure, pounds, 5.5–6.25.
Number of tuyeres, 7.

FIG. 10.—FURNACE No. 5.



Blown-in, Nov., 1869.
Blown-out, Apr., 1878.
Iron per week, tons, 182.86.
Average grade of iron, 4.34.
Fuel per ton pig, tons, 1.59.
Blast-temp., 650°–750°.
Blast-pressure, pounds, 4.5–5.
Number of tuyeres, 4.

FIG. 11.—FURNACE No. 2.



Blown-in, Aug. 22, 1869.
Blown-out, Feb. 6, 1871.
Iron per week, tons, 189.52.
Average grade of iron, 4.61.
Fuel per ton pig, tons, 48.
Blast-temp., 650°–750°.
Blast-pressure, pounds, 4.5–5.
Number of tuyeres, 5.

FIG. 12.—FURNACE No. 3.

No. 5 Furnace was blown-in in January, 1869, and at once showed marked superiority over the older furnaces, both in fuel-consumption and in regularity. In consequence, Nos. 2 and 3 were raised to 50 ft. in height, with the profiles shown in Figs. 11 and 12. At the same time, larger and better hot-blast ovens

were built, and the combined effect of these changes was a decided but not revolutionary improvement in the results.

Fig. 13 is of No. 2 Furnace at the Musconetcong works, Stanhope, N. J., built in 1869–70, following the good work of

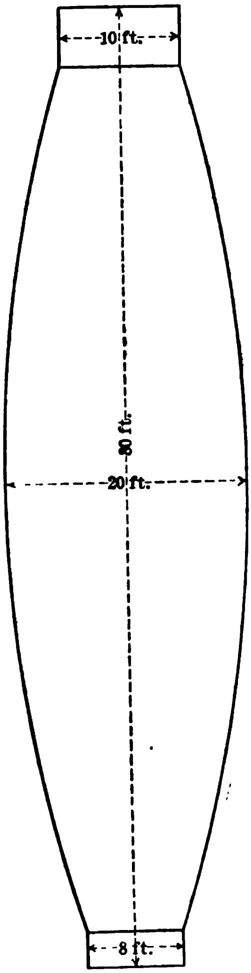
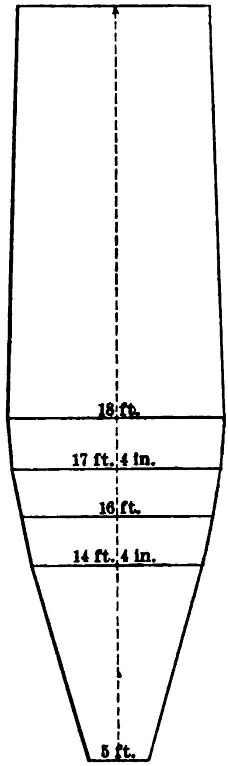


FIG. 13.—FURNACE NO. 2, MUSCONETCONG IRON WORKS, STANHOPE, N. J. 1871.



Blown-in, Jan., 1874.
Blown-out, Aug., 1874.
Iron per week, tons, 214.74.
Average grade of iron, 3.67.
Fuel per ton pig, tons, 1.52.
Blast-temp., 700°–800°.
Blast-pressure, pounds, 4.5–5.
Number of tuyeres, 5.

FIG. 14.—FURNACE No. 1.

No. 5 Glendon, and laid out on similar lines and the same proportions (H/d = 4). This furnace always surpassed No. 1 at the same works, even after the latter had been raised from 55 ft. to 75 ft. in height. I cannot give the exact figures as to output and fuel-consumption, but the latter was very good,

although rather higher than for No. 5 Glendon; the ore-mixture was different and generally not quite so rich as at Glendon.

In 1871, Furnaces Nos. 2, 3, and 5 were blown-out because of the coal-strike of that year, and advantage was taken of the opportunity to change them from open top and side flue to the cup-and-cone plan of closed tops. The cup-and-cone had been applied to the furnaces of the Bay State Iron Co. at Port Henry, on Lake Champlain, in 1865 or 1866, according to T. F. Witherbee.[13]

I have elsewhere described[14] the very unsatisfactory result at first obtained, caused by our ignorance of the proper proportion for the bells.

The '70's of the past century were years of great importance for the study of the blast-furnace process, marked as they were by the publication of Bell's great work[15] and that of Gruner,[16] and the many comments on them. In both works, special emphasis was laid on the importance of the action of the gas at comparatively low temperatures in the upper part of the furnace, which, as previously stated, had been announced long before by Gibbons.[17] All this knowledge naturally revived the schemes for furnaces with a great volume in the upper part, by giving experimental support to what had been previously more or less vague conjecture, and as the capital importance of the mode of filling had just been emphasized by the troubles attending the first use of the cup-and-cone, it was natural to attribute the ill-success with the domed section at Furnace No. 1 in 1850 and 1852 to the nearly-central mode of filling into the relatively small top. Acting on these notions, Furnace No. 1 was raised, in 1873, from 50 ft. to 63 ft. in height, and lined up to a greatest diameter of 18 ft., with a top diameter of 16 ft., as shown in Fig. 14. The filling was effected through six small bell-and-hoppers disposed in a circle in a large ribbed plate covering the entire top of the furnace. The gas was taken off by a pipe in the center of this plate, which at the same time supported by brackets the six air-cylinders which moved the bells.

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[13] *Trans.*, xxxviii., 894 (1908).

[14] *Trans.*, iv., 128 (1875–76); xiii., 520 (1884–85); and xxviii., 370 (1888).

[15] *Chemical Phenomena of Iron Smelting* (1872).

[16] *Études sur les Hauts-Fourneaux.*

[17] Percy, *ibid.*, p. 477 (1864).

This furnace was blown-in in January, 1874, and worked fairly well for some months, but showed no great improvement in fuel-consumption as compared with the work done before the alteration, and in this respect it was always far behind No. 5, which had practically the same cubic content, about 11,600 cu. ft. In July and August the furnace gradually worked less and less satisfactorily, and about August 9 it was almost completely scaffolded over, taking for some time only two or three charges per shift. Finally, it was blown-out when it appeared that, although nothing was left in it from the tuyeres to a height of about 15 ft. above them, at that level it was bridged over, the lower surface of the scaffold being nearly flat and level. The only opening through the scaffold, which was about 10 ft. thick, was a somewhat-tortuous hole 5 or 6 ft. in diameter next the walls on the right-hand side when facing the tymp. In trying to remove the scaffold quickly, water was so freely and incautiously used that the slaking of the lime in the suspended mass, by its expansion, completely wrecked the red-brick masonry, and no course remained but to tear the furnace down to the foundation.

This furnace departed so much from ordinary lines, both in shape and in method of filling, that it is practically useless to speculate on the reasons for its failure. Possibly, probably even, a much greater diameter at the tuyeres would have helped matters. Had we succeeded in clearing out the scaffold without injury to the furnace, as we could have done with more patience and discretion in the use of water, it would have been proper and desirable to make another trial, without altering the shape or method of filling; but during the short blast nothing had been observed to warrant the building of a new furnace on plans so different from those known to give at least good results. Accordingly, in rebuilding, the shape and dimensions shown in Fig. 15 were adopted, for it was not possible to build a duplicate of No. 5 without enlarging the foundations and tearing down adjoining buildings, which it was very desirable not to disturb. The red-brick work of the furnace was finished in November, 1875, but the lining was not put in until 1877. The furnace was blown-in in August, 1877, and gave excellent results, working regularly and with very low fuel-consumption, but, on the whole, no better than

No. 5, and perhaps not quite so well. Unluckily, direct comparison over a considerable period is not possible, because, during and after 1877, No. 5 ran much of the time on foundry-iron, and on a different ore-mixture from that used in No. 1. It never happened, however, when both were running at the same time on forge-iron, and therefore on identical ore-mixtures, that No. 5 did not do somewhat better as regards fuel-consumption than No. 1, and naturally, being considerably larger, No. 5 made rather more iron per week. John Fritz, in the extension of the Bethlehem Iron Works between 1870 and 1875, made the new furnaces 70 ft. high and 16 ft. in greatest diameter. These furnaces worked well in spite of a very variable iron-ore supply.

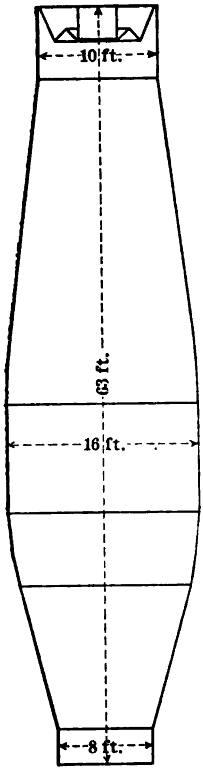
In 1879 it was decided to replace Nos. 2 and 3 by larger and higher furnaces, the advantages of which had been incontestibly shown by ten years' experience with No. 5. Drawings for this change had been prepared under my father's direction in 1876, which resulted, under the influence of Gruner's work and the unfavorable outcome at No. 1 in 1875, in a design for a furnace 80 ft. high and 18 ft. in greatest diameter. It differed from No. 5 in having the cylindrical part at place of greatest diameter, 6 ft. high instead of 3 ft., and the cylinder at the top 7 ft. high instead of 2 ft., as in No. 5. This design belongs to Gruner's *élancé* type, H/d being even greater than 4.

It is a good plan to have a cylinder of considerable height at the top, because it decreases the irregularities in distribution which arise when the furnace is not kept very exactly full to the proper stock-line.[18]

In 1879, in actually rebuilding the furnaces, beginning with No. 3, we had to consider, in addition to the experience previous to 1875, the excellent results during 1877–78 with the rebuilt Furnace No. 1, which had a top 10 ft. in diameter for a greatest diameter of 16 ft. It seemed probable that this form contributed to the good working, and that it would be well, therefore, to adopt a top diameter of 11 ft. for the new furnace, which is practically in the same proportion to 18 ft. as 10 ft. is to 16 feet.

The final design was for a furnace 81 ft. high, 18 ft. in greatest diameter, and 11 ft. in diameter at top, with the Coingt

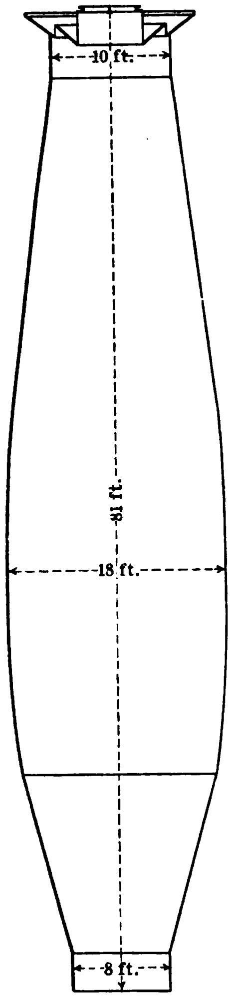
[18] Gjers, *Journal of the Iron and Steel Institute*, vol. iv., p. 211 (No. II., 1871).



Blown-in, Aug. 19, 1877.
Blown-out, Oct. 30, 1880.
Iron per week, tons, 304.2.
Average grade of iron, 3.13.
Fuel per ton pig, tons, 1.15.
Blast-temp., 800°–850°.
Blast-pressure, pounds, 6.5–7.5.
Number of tuyeres, 5.

Same profile, Langen charger:
Blown-in, Mar., 1881.
Blown-out, Apr., 1884.
Iron per week, tons, 340.2.
Average grade of iron, 3.01.
Fuel per ton pig, tons, 1.21.

FIG. 15.—FURNACE NO. 1, REBUILT 1875–77.



Blown-in, Aug. 5, 1882.
Blown-out, July 15, 1886.
Iron per week, tons, 412.4.
Average grade of iron, 2.75.
Fuel per ton pig, tons, 1.21.
(118 lb. coke.)
Blast-temp., 800°–850°.
Blast-pressure, pounds, 8–9.
Number of tuyeres, 7.

86 weeks unsized ore (1882–84):
Iron per week, tons, 349.5.
Average grade of iron, 2.53.
Fuel per ton pig, tons, 1.32.
118 weeks sized ore (1884–86).
Iron per week, tons, 458.3.
Average grade of iron, 2.88.
Fuel per ton pig, tons, 1.15.

FIG. 16.—FURNACE NO. 2, REBUILT 1881–82.

charger.[19] This furnace was blown-in in October, 1880. No illustration of it is given, since it differs from the new No. 2, Fig. 16, only in the diameter at the top, and the use of a Coingt instead of the Langen charger. In fact, No. 3 was altered to 10 ft. on top and the Langen charger substituted for the Coingt in May, 1882. The change in shape at the top was effected without blowing-out, by hanging a cast-iron liner, of proper shape, from the lining-ring. The change appreciably improved the working of the furnace.

By the time we were ready to rebuild Furnace No. 2, we had proved the advantages of the modified Langen charger,[20] and it was, of course, adopted for the rebuilt furnace. No. 2 was blown-in in July, 1882.

Neither of the rebuilt furnaces gave really satisfactory results, both showing decidedly greater tendency to scaffold and working irregularly than either No. 1 or No. 5, more especially No. 3, before the change to 10-ft. top and the Langen charger. Moreover, the necessary blast-pressure was decidedly higher, from 9 to 11 lb. per sq. in., as compared with from 6 to 8 lb. for Nos. 1 and 5. This latter disadvantage was greatly diminished by the use of one-eighth of coke, which we began soon after the blowing-in of No. 2. A small gain in regularity and an increased output also followed this change.

The new No. 3 was run for a considerable part of the time on foundry-iron, hence the figures for fuel-consumption and output are not properly comparable with those for Nos. 1 and 2, and are therefore not given.

It was certain that the inferior results at Nos. 2 and 3 were not due directly to the increased height, and any consequent increase in the crushing-strain on the materials, a matter about which much has been said and written, but, in fact, very little in the way of exact observation has been published, because, first, the actual height of the column of materials in the new furnaces, allowing for the height occupied by the charging-apparatus, was not sensibly greater than it had been in No. 5 during the time it was worked with open top; and second, No. 2 at the Musconetcong works, Fig. 13, always did better than No. 1 at the same works, No. 1 being 75 ft. high

[19] *Trans.*, xxviii., 370, Figs. 3, 4, and 5 (1898).
[20] *Trans.*, xiii., 520 (1884–85).

and No. 2, 80 ft. Moreover, there was a falling-off in the work of all the furnaces at Glendon, after the middle of 1879, which pointed pretty clearly to something which affected all of them in the same manner but in a different degree. The cause was finally discovered in a serious increase in the proportion of very fine magnetic ore from the mines in New Jersey, due to the incautious use of dynamite, and to great irregularity in the percentage of fine and coarse at different times, depending on whether at the time of shipment the stock of ore under the trestles at the mines was increasing or diminishing. By limiting the use of dynamite to narrow work in sinking and driving, which, by the way, did not sensibly increase the cost of the ore, the proportion of fine ore was much reduced, and by screening over ⅞-in. bar-screens and shipping the fine and coarse in separate cars, it was possible, within limits, to control the percentage of fine and coarse filled into the furnaces.

At the furnaces a charge was made up of fine or of coarse ore; the two sizes were never mixed in the same round. This was done, at first, because it was much easier to be sure that a certain number of charges of fine ore were filled per shift than that so many barrows in each and every charge during a shift were of fine ore; moreover, it insured that the fine was uniformly distributed around the circumference of the furnace, and not, perhaps, filled all on one side. The resulting improvement was surprising, as is shown from the data given in Fig. 16. It was far greater than could possibly arise merely from the decrease in the percentage of fine material. T. F. Witherbee noted long ago the desirability of regularity in the proportion of fine to coarse ore;[21] E. L. Uehling has strongly insisted on the advantages of sizing and charging each size by itself, and not mixed with another in the same round; and E. Belani has done the same.[22] I can emphatically confirm, from my own observation, all that these authorities say as to the resulting advantages.

The use of sized ore in the other furnaces wrought appreciable improvement, but it was not so remarkable as in Nos. 2 and 3.

In spite of the good result shown by No. 2, following the

[21] *Trans.*, iv., 377 (1875–76).

[22] *Stahl und Eisen*, vol. xxiii., No. 13, p. 777 (July 1, 1903).

use of a little coke, and especially after sizing the ore, the new furnaces must be regarded as decidedly inferior to Nos. 1 and 5, especially in regularity. The probable cause of this inferiority I do not pretend to indicate, but, considering the good results from No. 2 at Musconetcong, it may be in the too-great ratio of the diameter to the height, H/d being 4.5 in the Glendon furnace, as compared with H/d = 4 at Musconetcong.

The matter seems to show very well the possible cumulative effect of comparatively small errors; that is, in this case the ill-effect of an increase in the fine ore used was not very marked in Nos. 1 and 5, but was serious in Nos. 2 and 3, when to it was added the effect of a change for the worse in the profile.

In comparing the fuel-consumption and the average grade of the iron made after 1875 with that before that year, it is necessary to remember the revolution in the production of finished materials, especially of rails, which followed the development of the Bessemer process. Before 1875 there was always a great market for white-iron for the rail-mills, and it was often made intentionally; after that it was commonly necessary, as far as possible, to avoid producing it, and, in general, the iron used for puddling was much grayer than that previously employed. This, of course, tended to increase the average fuel-consumption at the furnaces and to decrease the output.

As regards the increased fuel-consumption of No. 1, after 1880, this was probably due, in part, to the very considerable increase in the weekly output, as compared with that from 1877 to 1879.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Modern Progress in Mining and Metallurgy in the Western United States.

PRESIDENTIAL ADDRESS.

BY DAVID W. BRUNTON, DENVER, COLO.

(Spokane Meeting, September, 1909.)

I. Introduction.

The list of our past-Presidents comprises the names of many who, in their official addresses, have sketched the current progress of the arts and professions with which they were familiar. Such addresses are, in my judgment, highly useful, setting forth the results of our own work as a society, and recalling to our minds the particular lines in which discussion would prove most fruitful. Following this course with some hesitation, I venture to offer an outline of the recent improvements and the present situation in mining and metallurgy in the Western United States.

The wonderful advances in mechanics, chemistry, and electricity have all combined to aid these arts to such an extent that progress during the last decade has undeniably been more rapid than ever before in the history of the profession. In all mining countries improved transportation-facilities, in many instances called into existence by the traffic created by the mines themselves, have done much to enlarge the field of operations by the reduction of freight-, operating- and living-expenses, thereby bringing lower-grade properties into the producing class. Under normal conditions, as the age of a district increases, all these different factors should combine to off-set the augmented costs attendant on deep mining, and greatly tend to prolong the profitable life of the mines.

The more recent and important improvements in our Western mining-practice which have contributed most toward the advancement of the art may be briefly summarized as follows; but, before beginning this portion of my address, I wish to thank most heartily the many engineering friends who have so

kindly furnished data covering their latest practice, without which this paper would have been even less complete than in its present form.

II. Mine-Mapping.

Not many years ago most mining companies thought it amply sufficient to have a surface-map of their properties and a composite map showing the different underground workings in their mines. To-day, almost every important concern maintains, in addition to the above, both stope- and assay-maps, while many of the larger companies add individual-level horizontal and vertical cross-section maps showing the underground geology in full. Upon these maps conventional designs in black ink are used to designate the various rocks, while the different veins or vein-systems are shown in colors. These sections are frequently drawn also upon glass sheets, which are then inserted in wooden frames provided with vertical or horizontal slots or grooves cut at the proper relative distances apart to correspond with any desired planes of cross-section or with the working-levels of the mines in question. The great advantages of such plans and sections cannot be overestimated. They not only show at a glance the tonnage and value of the ore in sight, but also afford a guide for development-work, whereas the old-fashioned maps were nothing but a record of the work performed, and were practically useless for any other purpose. The improvement named has brought about another, the importance of which is just beginning to be recognized. I refer to the employment by large mining companies of economic geologists, who are not burdened with the duties of surveying, directing workmen, etc., but give their whole attention to the geological problems encountered in the work. The advice of such experts in the purchase of property, the running of exploration-drifts, the location of shafts, etc., and the interpretation of local fault-systems, and other structural features, has already proved of inestimable value to their employers.

III. Surface-Mining.

Large ore-bodies occurring near the surface can, in many cases, be most cheaply and satisfactorily mined by stripping off the overburden and loading the mineral into cars by either the

"milling" or the steam-shovel system. In the West, steam-shovel mining is confined almost entirely to the low-grade copper-deposits at Ely, Nev., and Bingham, Utah. The system employed follows closely that of many iron-mines of Minnesota; and 95-ton shovels, with 3.5-cu. yd. (7-ton) dippers, are in common use. At Bingham, the Boston Consolidated Co. stripped the overburden from its deposit at the rate of 200,000 tons per month. The maximum amount handled in a calendar month was 282,903 tons, in August, 1907, and the maximum tonnage for a single day, with four shovels, is 15,000 tons. The Utah Copper Co., immediately adjoining this, is also carrying on equally extensive stripping and mining; and as many as 13 steam-shovels and 26 locomotives have been counted at work within a radius of half a mile.

IV. Rock-Drills.

The great improvements in core-drills, both diamond and calyx, enable us to-day to explore ground hundreds of feet in advance of the actual openings and afford great aid in all development-work. Power-drilling has now almost entirely replaced hand-work, and a vast assortment of drills has been placed on the market, from which a careful engineer will often have extreme difficulty in selecting the machine best adapted for a particular service. Rock-drills, reciprocating, air-hammer, and electric-air, are all in successful operation to-day; and with the steady improvement, both in design and material employed in construction, there is every reason to believe that the drill of the near future will be even more nearly perfect than those now in use.

V. Mine-Hoisting.

Thirty years ago, while immense hoisting-plants were in use on the Comstock, they were far from efficient, and were not copied even in miniature on smaller mines. The favorite plant in Colorado in the early days was called the Gilpin county hoist, and consisted of a rope-drum securely fastened at one end to a large wooden pulley connected by a slack belt to a stationary engine running continuously at a slow speed. When the signal was given to hoist, the operator opened the throttle of the single-cylinder engine and brought the necessary work-

ing-tension on the belt by means of a tightener operated by a hand-lever. To-day, these primitive machines, large and small, have all been succeeded by direct- or gear-connected steam-engines, equipped, whenever the tonnage is sufficient to justify the expense, with variable cut-off valve-gear, post-brakes, and every modern improvement.

Steam-hoists, capable of handling from 10 to 20 tons of total load, from depths of from 4,000 to 6,000 ft., at speeds varying from 4,000 to 5,000 ft. per minute, are now not at all uncommon. These immense plants are fitted with every imaginable device for increasing the efficiency, rapidity, and safety of operation, and the skill and artistic ability displayed in the design of some of the later Nordberg creations bring them to a point where they can almost be considered works of art.

In many places where water fit for use in boilers is scarce and electric current cheap, as at Cripple Creek, the electric hoist has almost completely replaced the steam-hoist.

When large electric hoisting-plants were first installed, it was found that the great amount of current necessary to start and accelerate the load brought a very objectionable "peak" on the transmission-line. This difficulty has now been overcome by the Illgner and other similar systems, in which the energy stored in a large rapidly-revolving fly-wheel cuts down, if it does not entirely prevent, the objectionable peak. Safety-devices likewise have been very much improved; and a recent invention, whereby the cage-tender is in constant signal-connection with the engineer, should do much to decrease the number of shaft-accidents.

VI. Underground Tramming.

When mines were shallow, shafts numerous and hoisting-facilities inadequate, hand-tramming was almost universally employed, but with increase in depth came the necessity for better hoisting-machinery and a reduction in the number of shafts, thereby increasing the distance over which ore had to be trammed. Then it was found that a high-priced man constituted a very expensive motive-power for pushing cars; horses and mules were put into commission; and, still later, air- and electric locomotives have come into very general use. There has been much discussion concerning the relative merits of

these two latter systems of underground haulage; but there is no doubt that each has its own field. Where the openings are dry and the roof sufficiently high and firm to carry the trolley-wire insulators, there is no question as to the desirability of using electricity, but where these conditions unfortunately do not obtain, the compressed-air locomotive is an excellent substitute.

VII. Timbering.

Where the ore-bodies do not exceed 10 or 12 ft. in thickness, and have a firm hanging-wall, nothing can exceed the cheapness and simplicity of stulls; but when larger ore-bodies are encountered and timbering is necessary, the system commonly employed is that of "square setting," invented by Deidesheimer and first used on the Ophir mine on the Comstock lode in 1861. Timber is yearly becoming more expensive, does not usually last well underground, and when the ore-bodies are large, especially if there is a tendency to movement in the walls, "square sets" made from square timbers are apt to "swing" and afford very little vertical support.

When large quantities of timber are required for square setting, in situations where the distance from the forest to the mine is not too great, round timbers are very much cheaper and more durable than square. A round log has about double the strength of a square timber cut from the inscribed square on its small end; and since, in the round log, the concentric rings of wood-growth are unbroken and each protects the ring immediately underneath it from decay, the comparison, both in cost and in durability, is very unfavorable to square timbers. Automatic framing-machines can now be had which utilize the full strength of round timbers by making a bevel-joint outside of the square tenon necessary in all square-set timbering. This additional segmental contact-area in the joints braces the round timbers so that they are much less liable to "swing" in large stopes than the square.

In some cases, "cut and slice" and "caving" methods are employed, in which the hanging-wall is allowed to come down and rest on each successive floor as the ore is stoped out, and is prevented from mixing with the ore by a mass of crushed timbers and plank which follows down on the top of the receding ore. In the Utah Copper and Boston Consolidated mines,

at Bingham, Utah, about 4,000 tons of copper-ore are mined daily by the "caving" system. In some large mines the stopes are filled with waste as fast as they are freed from ore, and the ground above thereby prevented from caving, in the same way as if stulls or square setting were used. Steel and concrete are coming slowly into use in shafts, stations, and tunnels, and with the natural decrease in the price of iron and cement on the one hand and the rising cost of timber on the other, it is easy to see that the more durable forms of construction will eventually supersede wood on all permanent work.

The direct-replacement system employed in the Rio Tinto copper-mines in Spain succeeded perfectly in holding both walls and surface in place on a vein from 200 to 260 ft. wide, and, by taking advantage of the wonderful skill of the Spanish miners in building dry stone walls, gives a new method of safely and economically mining the lower portions of the lodes which cannot be reached by the open-cast systems extensively in vogue there.

VIII. Pumping.

In the United States, the old-fashioned Cornish pump, with its costly foundation, massive walking-beam, huge plunger-rods and ponderous balance-bob, was supplanted many years ago by the direct-connected steam-pump, which soon developed into a most efficient pumping-machine with duplex, triple-expansion engines and every refinement possible in modern steam-engine practice. These have in many places been superseded by the electric-driven plunger-pump, in which the high speed of the electric motor has been reduced by suitable train-gear. Latterly, quite a large number of electric pumps have been built in which the gear is entirely eliminated. The speed of the motor has been reduced, and that of the plunger raised, forming a combination known as the express pump. Pumps of this class, with capacities of 1,600 gal. per minute, raising water 1,550 ft., are now being very successfully employed in unwatering the Comstock Lode at Virginia City.

Within the last few years great improvements have been made in the electric-driven turbine, which, with its entire absence of valves and reciprocating parts, threatens to dominate the field completely.

Already we have single-stage turbine-pumps raising 35,000

gal. of water per minute 150 ft. high; five-stage pumps raising 10,000 gal. per minute 600 ft.; six-stage pumps raising against 800 ft. head; and eight-stage pumps raising 400 gal. 1,400 ft.; and responsible firms are ready to contract to raise water by this system to any elevation up to 2,000 ft., and guarantee a pump-efficiency of from 60 to 75 per cent., according to conditions of service.

IX. Mine-Lighting and Signaling.

Incandescent electric lighting has long since driven oil-lamps and candles out of underground stations and permanent levels in which any large amount of work is carried on. The recently invented tugsten-lamp with its high efficiency, giving 20 c.-p. with an expenditure of only 25 watts per hour, makes it economically possible to extend electric lighting very greatly throughout underground workings. In some of the largest and most progressive mines, candles and oil-lamps have already been replaced in the stopes by acetylene-lamps, which are not only cleaner and safer, but give a much greater illumination for a given cost.

For mine-signaling, the flash-light system operated by interrupting, by means of well-protected switches, the current passing through the station-lamps is rapidly replacing the old-fashioned cumbersome bell-cord; and the latest moisture-proof mine-telephones give instant communication throughout the underground workings, and to and from the surface, with little danger of interruption.

X. Explosives.

The use of modern high explosives in mining and tunneling is now universal; but there is a crying demand for an explosive which can be more safely handled, and which on explosion or detonation will produce a smaller amount of noxious gases, which not only injure the health of the miners, but delay the resumption of work after each round of shots has been fired.

Irregularities in the composition of explosives, variations in the strength of detonators, and differences in the speed of fuses are all fruitful sources of mine-accidents; and, while too much "paternalism" is certainly to be avoided, it is doubtful if anything short of government regulation and inspection of explosives, detonators, and fuses will ever bring about the uniformity necessary to safety.

XI. Mine-Ventilation.

Less progress has taken place in this department than in almost any other, although the means for moving large quantities of air under slight pressures have been very much improved. The ventilation and cooling of metal-mines have not yet received the attention which their importance demands. In this respect Western engineers could take profitable object-lessons from their brethren in the coal-fields. Very few of our Western mine-operators go to the trouble of recording temperatures and making ventilation-maps, showing the direction of the air-currents, etc., all of which data are necessary before a satisfactory system of either natural or artificial ventilation can be planned. As most of the Western mines are in hilly or mountainous situations, it is generally easy to provide two openings at greatly different elevations, so that the heating-effect of the workings can be depended upon to control the direction of the air-currents to such an extent as at least to cool and ventilate the workings partly. When these advantages cannot be obtained, centrifugal or forced-draft blowers, driven either by steam or by electricity, furnish an easy means of obtaining the desired results.

The latest high-speed electric direct-driven centrifugal compressors give pressures up to 45 oz., and have been built in sizes up to 40,000 cu. ft. of free air per minute; but there is apparently no limit to the size of the machines which can be built under this system.

XII. Tunneling.

As the United States continues to grow in wealth and importance, tunneling operations increase in like proportion, both in number and in magnitude. New York City and its environs are now underlain by a net-work of tunnels, and other cities are rapidly developing underground systems, since through the increase in population business and travel become congested. All over the United States, water-supply, hydro-electric power and the reduction of grades on railways are requiring new and expensive tunnels, to which, in the West, are added the great irrigation-tunnels called for in both government and private enterprises. The result of all this activity in tunneling has been a vast improvement in both machinery and methods, and

a greatly increased number of thoroughly trained and skilled workmen, so that records formerly unattainable in the United States are now being made in widely-separated localities. Table I. gives the monthly rates of progress reached by some of our latest achievements in tunnel-driving. The recent paper of Mr. Saunders[1] describes systems and results abroad, which show much higher rates of progress than we have yet been able to attain here.

TABLE I.—*Progress in Driving Tunnels in the United States.*

Cowenhoven tunnel, Aspen, Colo., 7 by 8 ft.,	May, 1893, 421 ft.
Roosevelt tunnel, Cripple Creek, Colo., 9 by 10 ft.,	January, 1909, 435 ft.
Gunnison tunnel, Gunnison, Colo., 12 by 12 ft.,	January, 1908, 449 ft.
Elizabeth tunnel, Los Angeles, Cal., 12 by 12 ft.,	October, 1908, 466 ft.

XIII.—GOLD-DREDGING.

Chain-bucket dredging for gold was first attempted in 1867 in Otago, New Zealand, and the first steam-actuated dredge operating on this principle was built on the Molyneux in 1881.

From a few small dredges copied after those in use in New Zealand, gold-dredging in this country has grown into a great industry, which is carried on successfully from the frozen gravels of the Arctic to the sun-scorched river-bars of the tropics, and at all altitudes from 10,000 ft. down to sea-level. Under stress of competition and the necessity of meeting new conditions, dredges have grown both in capacity and efficiency to a point not even dreamt of a few years ago. Dredges are now built with close-connected buckets, of capacities up to 13.5 cu. ft., and capable of handling 10,000 cu. yd. of gravel in 24 hr.; some have been built with bucket-ladders capable of digging 67 ft. below the water-line and 20 ft. above it. A new stacker now under construction will deliver tailings 160 ft. away from and 60 ft. above the deck of the boat. Improved construction and better management have rendered dredging-operations less dependent upon weather; and last winter in Colorado a dredge was operated continuously at an altitude of 9,990 ft. where the temperature on several occasions fell to 20° below zero. The subject of suction-dredges has been fully considered in Mr. Granger's recent paper.[2]

[1] *Bulletin* No. 28, April, 1909, pp. 337 to 364.

[2] *Bulletin* No. 28, April, 1909, pp. 389 to 410.

XIV. Electric Transmission.

No sketch of this kind would be complete without some notice of the immense service which the mining-industry is receiving from long-distance electric transmission. While a few mines are favorably situated for the utilization of adjacent water-power, many of the principal mining-districts of the United States are at altitudes so great that any available water-power is far below them. Again, as in the case of Nevada, Arizona, and portions of Utah, the mines occur in an arid country where it is difficult to obtain sufficient water for domestic purposes, to say nothing of power.

Already the electric current is carried to all elevations from sea-level to timber-line, and there is scarcely a desert mining-camp of sufficient size to justify the erection of a pole-line that is not equipped with electric power. The ease with which this overcomes the old and apparently insurmountable problems of scarcity of water and fuel constitutes one of the delights of modern mining.

The use of electricity has also completely solved the old vexed problem of underground sinking and hoisting, so that these operations are now as readily carried on from deep tunnel-levels as from the surface.

Transmission-lines of all lengths up to 220 miles are in daily use, with carrying-capacities ranging up to 40,000 kw. Long-distance transmission-systems are in many cases operated at 100,000 volts, and new lines are building to utilize even higher pressures. Recent improvements in insulation promise to make still higher voltages possible, which would mean a corresponding increase in the distance to which current could be profitably carried.

XV. Sampling.

Few departments of mining engineering have shown greater advance than ore-valuation.

In milling- and concentrating-plants, where fine crushing is a necessary preliminary, sampling is a comparatively easy and reasonably accurate operation, but where the ore is to be treated by blast-furnace smelting, crushing of any kind is objectionable and fine subdivision is prohibited. Forty years ago, for the valuation of coarse ore, "grab" sampling was in common use, and this method was replaced in slow succession by Cornish

quartering, fractional division, and split-shovel sampling. Then came automatic sampling in many forms, but all taking a portion of the ore-stream continuously. In 1884 a new system of sampling was invented which automatically deflects the entire ore-stream for a varying portion (usually one-fifth) of the time into the sample-division. Numerous different machines working on this principle are now in use; and these types of sampling-plants have been perfected to such a degree that where the hopper ore-cars which are now coming into general use are employed, ore may be unloaded, crushed, sampled, and reloaded into the outgoing cars, and the ground sample delivered in a locked steel box, without ever having been handled—the entire chain of operations being performed automatically.

XVI. Concentration.

The separation of valuable minerals from worthless gangue must have been one of the earliest operations in the history of metallurgy. Up to less than 100 years ago the pan, tub, and inclined plane, which are all so graphically illustrated by Agricola, continued to be the only devices in use. Hand-jigs were first introduced for the separation of coarser particles than could otherwise be handled, and the principles involved are in use to-day, although improved mechanical appliances have changed and enlarged operations to such an extent that the primitive origin would scarcely be recognized. About 35 years ago the use of air as a concentrating medium was successfully introduced, and, despite its many disadvantages, this system, assisted by numerous mechanical improvements, still exists and manages to hold its own where water is unobtainable or, for some reason, cannot be used. In skillful hands, some of the pneumatic separators give wonderful results, but the delicacy of the adjustments and the attendant dust will undoubtedly prevent any extensive employment of this method.

The pulsating water-current recently invented by a distinguished investigator in the concentrating field has already won a place for itself in both sizing- and jigging-operations, and promises to become a most important factor in concentrating-work.

Specific gravity is, however, no longer the only principle taken advantage of in mechanical ore-sorting. To these have

been added magnetic and static electric separation, and many different methods based on the surface-tension of water (with or without the assistance of oil or acid), resiliency, and affinity for grease. The latter method, used very sparingly in this country, finds its chief application in South Africa, where it is used in the separation of diamonds from other stones.

The recent discovery by Kunz and Baskerville that the use of ultra-violet light would enable an observer to determine by inspection, with reasonable accuracy, the percentage of willemite in concentration-tailings, has already found commercial application on a very large scale, and opens up a wide field for speculation as to what the future may hold in store for us in this field.

The enormous size of some of the new concentrating-plants erected in the West exemplifies in a marked degree the magnitude of the operations now being carried on. At Anaconda, Mont., the Amalgamated Copper Co. has an eight-unit concentrating-plant, each section of which handles 1,000 tons in 24 hr.; and some of the new plants at Ely, Nev., and Garfield, Utah, are but little smaller in size.

XVII. Roasting for Blast-Furnace Smelting.

The earliest roasting-furnaces to prepare sulphide ores for blast-furnace smelting were small hand-operated reverberatories, with or without fusion-hearths. These were followed by revolving cylinders and various types of mechanically operated reverberatory furnaces, all of which were not only expensive to operate and keep in repair, but yielded a product very badly adapted to blast-furnace work. To-day these old-fashioned furnaces, both hand and mechanical, have been almost entirely superseded by blast-systems like the Huntington-Heberlein, Carmichael-Bradford, and Savelsberg, which make it possible to utilize the many advantages of blast-furnace smelting for the treatment of concentrates, fine ore, and flue-dust at very low preparatory costs. These systems marked a wonderful advance over the old reverberatory roast with its pulverulent product; but they were still open to the serious objection of requiring a large amount of manual labor in charging and discharging the pots and breaking up the sintered product. In spite of this drawback, the results obtained were so desirable that study and in-

vention along these lines have been stimulated to such an extent that there are already in use mechanical roasting- and sintering-plants (system of Dwight and Lloyd), to which ore can be fed in a steady stream, and which will automatically deliver a desulphurized, sintered, and broken-up product in the best possible condition for blast-furnace work.

XVIII. Lead-Smelting.

Progress in this department of metallurgy during the past decade, in the West, has been much hampered by three conditions: (1) all the lead-smelting plants operate almost entirely on custom ores; hence the supply is irregular in volume, grade, and composition; (2) the great extension of the leasing system throughout the West tends to bring ore into the market in very small lots, thereby increasing the difficulties and cost of storage and bedding; (3) eight years ago nearly all of the principal lead-smelting plants in the United States passed into the hands of a corporation organized for that purpose, and the industry was thereby deprived of the stimulus of healthy competition.

The largest lead-furnaces in the United States have hearths 180 by 44 in., and treat daily from 150 to 255 tons of ore, according to its character. Mechanical charging is used in some cases; but while it slightly reduces operating-costs, it is no improvement metallurgically.

In Australia, where different conditions prevail, the improvements in lead-furnaces have kept pace with those in iron- and copper-smelting.

XIX. Reverberatory Copper-Smelting.

In 1867, when Richard Pearce (afterwards President of the Institute) built his first reverberatory furnaces at Black Hawk, Colo., they were considered the acme of metallurgical perfection, and their successful operation did wonders for the mining-industry of the State. The hearths of these furnaces were only 8 by 12 ft. in size, and their daily capacity was 12 tons. As the quantity of ore produced increased and the necessity for handling larger tonnages became apparent, the reverberatory furnaces have been steadily enlarged and improved, until to-day they have attained almost incredible dimensions,

having a hearth-area of 19 by 116 ft. and a daily working-capacity of more than 300 tons, which, in the case of easily-smelted ores, has risen to over 400 tons. These large furnaces secure great saving of heat and uniform, continuous operation, for reasons into which I need not enter here.

XX. Blast-Furnace Copper-Smelting.

The early water-jacketed blast-furnaces for smelting copper-ore were small, round, wrought-iron affairs, about 30 in. in diameter and rarely smelted more than 12 tons of ore in 24 hr. From this puny beginning, keeping pace with the rapidly-growing copper-industry of this country, furnaces have increased steadily in size and improved in mechanical construction, until they have reached the enormous dimensions of 87 ft. in length by 4 ft. 8 in. in width, with a daily smelting-capacity of 3,000 tons of charge. These furnaces are mechanically fed, work under an air-pressure of 40 oz., and give infinitely less trouble than their smaller progenitors.

The water-jackets are completely sectionized, and it is possible to renew most of the sections without stopping the furnace. At first only one tier of jackets was used, then two tiers came into use, and now some of the most recent furnaces have air- or water-jackets replacing the brick superstructure, thus doing away with much of the roof-accretion nuisance. On furnaces built with a crucible, baby water-jackets have replaced the old cast-iron plates, and water-jacketed nose-pieces have greatly lengthened the life of the furnace-discharge spouts.

The credit for the latest improvements in design and increase in size of both reverberatory and blast-furnaces is due principally to E. P. Mathewson, whose untiring energy as an investigator and skill as a metallurgist have made the Washoe plant, at Anaconda, the Mecca for progressive engineers from every country.

The recently invented acetylene blow-pipe promises to be of great service to blast-furnace engineers, as through its use it will soon be possible to obtain welded water-jackets entirely free from the objectionable lap-seams and rivets.

XXI. Electrical Smelting.

A great amount of experimental work has lately been carried on by engineers in various parts of the world having for its

object the utilization of the electric current in metallurgical operations, and there is no doubt that in the near future many minerals now smelted with fuel will be reduced to metals by electrical processes. The electric current possesses the great advantage of allowing a most efficient utilization of the heat; and also complete control of the exposure of the molten metals to air or gases.

XXII. Briquetting.

At many blast-furnace plants "fines" are made into briquettes with the ordinary die-and-plunger machines; but if free acid or copper sulphate be present, the surfaces of both dies and plungers are rapidly corroded, which obviously increases the diameter of the dies and diminishes that of the plungers. As this solvent action continues, a point is soon reached when the plunger no longer fills the die-opening, and pressure forces the material through the space between them, instead of consolidating the mass.

When a sufficient amount of plastic material, such as slimes, can be obtained to mix with fine ore and flue-dust, so as to give the mass the property of "flowage" under pressure, it may be briquetted in machines similar to those used in making building-brick by the "stiff-tempered" process. Solvents do not interfere with the operation of these machines, and by constructing the working-parts of steel and phosphor-bronze, it is possible to make briquettes dry enough to pass directly into a blast-furnace at the rate of 600 to 800 tons per day for each machine employed, at a cost of less than half that of the die-and-plunger system.

XXIII. Chlorination.

Chlorination is confined almost entirely to sulphide gold-ores carrying so little silver that its loss may be disregarded, and which require a preliminary roasting before treatment. The early methods of tank-leaching have all been superseded by barrel-chlorination, and in some of the latest plants chlorine is produced by electrolysis instead of by the decomposition of bleaching-powder with sulphuric acid.

The largest plants operating under this system treat custom-ores, and the stress of competition, together with the necessity of handling constantly increasing tonnages, has brought about great improvements in both machinery and methods.

The most desirable features of this process of gold-extraction are the high percentage recoverable, and the rapidity with which clean-ups can be made, rendering it easy at all times to know exactly what results are being obtained.

The largest plants operating under this system are situated in Colorado City, Colo., where two mills owned by one concern have an aggregate capacity of 800 tons per day.

XXIV. Cyanidation.

While the first patent for extracting gold from its ores by cyanide solutions was issued in 1867, it was not until McArthur and Forest took it up in 1889 that practical results of any value were obtained. Since that time the use of the process has increased by leaps and bounds in all of the principal gold-producing countries, and to-day it is the principal factor in the world's steadily increasing gold-production. Cyaniding seems to work with equal facility on raw ore, roasted ore, and tailings, and with the steady improvement in the mechanics as well as the chemistry of the process, it bids fair to do even greater things in the future than it has done in the past. Nor is its use confined entirely to the extraction of gold. In many districts it is operating with great success on mixed gold- and silver-ores. At Millers, Nev., two plants with an aggregate capacity of 700 tons per day are operating on Tonopah ores, in which the average ratio of silver to gold is 80 to 1.

The largest and perhaps the most complex cyanide-plant in the United States is the Golden Cycle mill, at Colorado City, Colo., which has a daily capacity of 1,000 tons, and treats exclusively Cripple Creek ore, all of which requires careful roasting and very fine crushing. The Homestake cyanide mill handles a larger tonnage, but treats only tailings.

Next to this plant in point of size and importance, and handling ore of much higher grade, is the new mill of the Goldfield Consolidated Co., Nevada, which has a capacity of 600 tons per day, and contains the very latest improvements, culled from American, African, Australian, and Mexican practice.

XXV. Fume-Recovery.

No department of metallurgy has made slower progress than this most important division, but now, through an unholy alli-

ance between the unscrupulous contingent-fee attorney and the greedy land-owner, the success of "smoke-farming" is compelling the smelting companies to do for self-preservation something which they should long ago have undertaken for profit. Out of the almost numberless devices which have been tried for fume-recovery, the bag-house affords the best solution of the problem yet devised. Bags were used for the recovery of both zinc oxide and lampblack more than fifty years ago, but were not used for fume-recovery until 1878, when Bartlett set up a small plant at Portland, Me. The first large successful installation of this kind was erected by the Globe Smelting Co., in 1885, and has been in continuous operation ever since. While the bag-house has been eminently successful in the recovery of blast-furnace fume, it cannot be used on fumes from reverberatory roasting- or smelting-furnaces, owing to the fact that a portion of the sulphur dioxide formed by the oxidation of the sulphur is raised to sulphur trioxide by contact with incandescent ferric oxide. At the United States Smelting Works, in Utah, provision for protecting the bag-house from sulphur trioxide is made by blowing into the flues zinc oxide, which immediately absorbs the sulphur trioxide present to such an extent that it has been found quite possible to use cotton bags. The bag-house has proved very efficient in recovering fume from lead-refineries and all lead-smelting operations excepting roasting. In large copper-smelting plants, where large volumes of gas are produced carrying a sufficient amount of sulphur trioxide to destroy woven fabrics, three systems are in use—namely: radiation, decreased velocity, and friction—and in many cases two or more of these methods are combined. At the Washoe plant, in Anaconda, long steel-covered flues of enormous cross-section have been installed for a number of years, and, while this system does not effect a complete recovery of the fumes, it is as nearly perfect as it is possible to make a plant to-day. At Great Falls, Mont., the Boston & Montana Co. is installing the friction system at a cost exceeding $1,000,000, which includes the construction of a stack 506 ft. high and 56 ft. in diameter and a dust-chamber in the flue-system, of such width that the furnace-gases will pass through it at a velocity considerably less than 500 ft. per minute. From the roof of this flue-chamber more than a million steel wires will

be suspended, an arrangement which experiment has shown to increase greatly the settling-efficiency of the dust-chamber. This installation is practically completed, but it will be some time before the results obtained can be accurately determined.

I need not add that where (as in some Eastern works) metallurgical and commercial conditions permit the manufacture of suphuric acid from the fumes, this method offers special advantages.

XXVI. Conclusion.

The subjects already alluded to occupy but a small portion of the field covered by our 4,000 widely scattered members, but I hope that enough have been mentioned to serve as topics for discussion at this meeting. The ever-widening range of operations, the constantly expanding magnitude of mining undertakings, and the continually increasing complexity of both machinery and methods are daily creating new openings for mining engineers. To meet this demand our technical schools and colleges are yearly sending out an increasing number of graduates, whose opportunitites and responsibilities will be even greater than those of the engineers controlling the activities of to-day. Even now, one change very much to be desired is beginning to become apparent. Heretofore it has too often been considered that an engineer's accountability ended when he discharged his full duty to his employer. To-day we are beginning to realize that the public forms a third party, vitally concerned in the results of the work in which mining engineers are engaged. As large investments are usually held by divided ownership and stocks are often scattered far and wide, so that the owners of small holdings have little or no opportunity to become conversant with the exact conditions of the properties they represent, an engineer's first duty should be to see that no word or act of his can be construed so as to give one man an opportunity to take advantage of, or mislead, another. Every one, no matter what his station, has a duty to society and his fellow-men which can never be either ignored or neglected. The employer, whether an individual or a corporation, is entitled to all of the information and data which experience, diligent investigation, and careful study can bring to light, and while an engineer has a right to state probabilities from both

indications and analogy, he should never assume the gift of prophecy and thereby delude both himself and others.

Specific information gained in examination or research for one client should never be utilized for the benefit of another, unless there is no possibility that such use will in any wise injuriously affect the interests of any previous employer. This restriction applies with full force to the dealings of the engineer himself, as he should always remember that information gained by him at the cost of another, no matter how laboriously it has been obtained, belongs to the party who paid for it. No matter what success, ability, industry, or chance may bring to an engineer, his career has been an absolute failure unless he can truthfully say in his heart of hearts: "No man is poorer because I am richer."

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Influence of Bismuth on Wire-Bar Copper.*

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(Spokane Meeting, September, 1909.)

Introduction.

THIS study was undertaken on account of the lack of definite knowledge concerning the influence of bismuth on wire-bar copper, and the small elimination of bismuth from copper-matte during the smelting-operations.

The early workers who studied the influence of bismuth on copper confined their investigations to malleability and ductility—two physical properties which are related to the others so intimately that their determination is of primary importance. Karsten [1] and Levol,[2] using bending- and mealleability-tests, seem to have been the first to take up the subject, and while their results are incomplete, they agree in the main with the later work of Hampe.[3]

The bending-tests used by the first investigators consisted in bending the specimen until it failed. The malleability-tests were made by beating the material to a knife-edge with a hammer. Hampe not only determined the percentage of bismuth which would cause red-shortness and cold-shortness of the alloy, together with other influences of bismuth on the malleability and ductility of copper, but also took up the influence of the bismuth protoxide, BiO, on copper, and of bismuth sesquioxide, Bi_2O_3, on the copper protoxide, CuO. Hampe's conclusions of the influences of these two latter compounds of bismuth on copper are as follows:

"If the protoxide of bismuth be alloyed with metallic copper, it is not changed to metallic bismuth, but remains mechanically distributed throughout the copper.

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[1] *Zeitschrift für das Berg- Hütten- und Salinen-Wesen im Preussischen Staate*, vol. xxii., pp. 93 to 138 (1874).

[2] *Ibid.*

[3] *Ibid.*

In this state the effect of bismuth seems to be less disadvantageous than in the metallic condition. The difference, however, is insignificant and restricted only to ductility, and then when cold. A material diminution in cold-shortness is shown, if the oxide of bismuth is combined with the protoxide of copper. Such alloys are far less cold-short than those containing the same amount of bismuth in the metallic form."

The Alloys Research Committee found a drop in tensile strength of 11,000 lb. per sq. in. in comparing two copper-alloys; one containing 0.10 and the other 0.20 per cent. of bismuth.[4] This result was the reason for including in their work the determination of the melting-point of the alloys. It was beyond comprehension why bismuth, which occurs in the Periodic Law next to arsenic and antimony, should diminish the tensile strength, while both arsenic and antimony have a tendency to increase it.

In order to study this strange influence of bismuth, Arnold and Jefferson[5] took up the subject from the stand-point of microstructure, and their conclusions corroborate the results of the Alloys Research Committee in their melting-point determinations. The results obtained by E. A. Lewis[6] agree with those of Hampe on malleability, and of Arnold and Jefferson on the structure of the alloy. Moreover, Lewis ascertained the influence of bismuth on copper, which contained also arsenic, tin, manganese, and aluminum individually. His conclusion is that "the injurious influence of bismuth is offset by the presence of arsenic, and intensified by tin, manganese, and aluminum."

Inasmuch as but little work has been done on tensile tests of copper-bismuth alloys, I decided to test the malleability and tensile strength of a series of copper-bismuth alloys, cast into bars, supplementing the work with similar tests on the same specimens passed through rolls, in order to determine the effect of rolling.

Recently, A. H. Hiorns[7] has published the conclusions of his

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[4] *Proceedings of the Institution of Mechanical Engineers* (London), Part 2, p. 120 (Apr., 1893).

[5] *Engineering*, vol. lxi., p. 176 (Feb. 7, 1896).

[6] *Journal of the Society of Chemical Industry*, vol. xxii., No. 24, p. 1351 (Dec. 31, 1903).

[7] *Journal of the Society of Chemical Industry*, vol. xxiv., No. 9, p. 501 (May 15, 1905).

comprehensive investigation of the microstructure of copper-bismuth alloys.

With regard to the elimination of bismuth from copper-matte, Edward Keller[8] found that in refining the matte 54 per cent. of the bismuth was eliminated in the reverberatory furnace, while 95 per cent. was eliminated in the converter. Even though the converter process be used, the percentage of bismuth remaining in the refined copper is large, since a 95 per cent. elimination is lower than the extraction of arsenic, antimony, and sulphur, which is 98 per cent. and more.

Raw Materials.

The metals used in my investigation were purest wire-bar copper, obtained in ingot form from the Nichols Chemical Co., Laurel Hill, Brooklyn, N. Y., and metallic bismuth, c.p., supplied by Eimer & Amend. The bismuth was added directly to the molten copper instead of diffusing it in the main mass by first alloying it with a small amount of copper, as had formerly been the practice. The pure elements of the alloy were used in preference to copper of known bismuth-content for two main reasons: (1) the difficulty of procuring samples containing the exact percentage of bismuth best adapted to the research; and (2) the probable presence of other impurities which would interfere with the tests. As a check, a careful chemical analysis was made of one of the series prepared synthetically. The alloy, prepared to contain 0.20 per cent. of bismuth, gave on analysis 0.18 per cent. In the alloys containing a smaller proportion of bismuth the error was probably less than 10 per cent.

Casting the Test Bars.

For the first melts on pure copper the ingot of wire-bar copper was cut by a hack-saw into cakes, each weighing about 3.5 lb. The cakes were melted individually in a No. 6 Dixon graphite crucible, previously heated to a bright red.

The melt, made in a No. 2 melting-pot of the American Gas Furnace Co., took from 20 to 30 min. During the melt the mold was heated over a gas-tube furnace, and was inclined by means of an iron plate, the gate end being the lower. Details of the construction of the mold are given in Fig. 1.

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[8] *Trans.*, xxviii., 157 (1898).

A movable clay-cover was placed on the crucible so that it could easily be removed, both to determine the "pitch" and to facilitate the addition of the bismuth. The desired amount of bismuth was carefully weighed, placed in a capsule, dropped into the molten copper, and rapidly stirred with a clean red-hot iron rod. As soon as the copper in the crucible was melted the lid was removed from the melting-pot, and a portion of the metal in the crucible was dipped out with a small ladle, previously heated to a red heat so as to prevent chilling and sticking. This sample was allowed to cool and set. If the surface rose, the metal contained too much carbon monoxide; if it

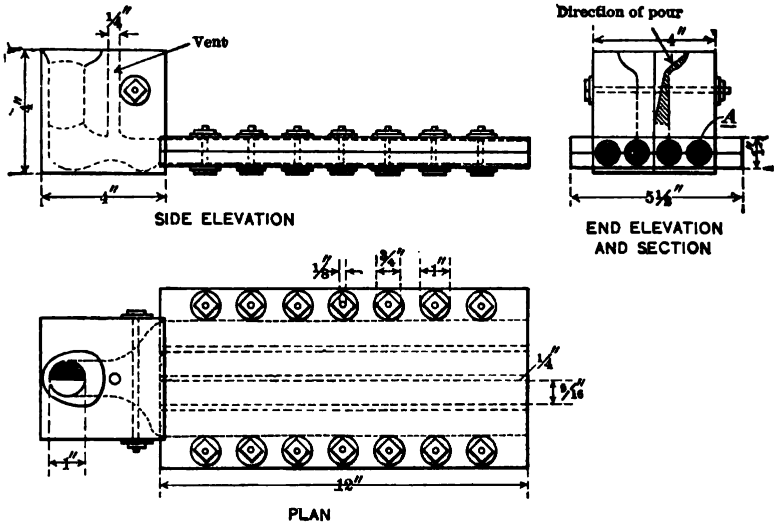

Fig. 1.—Detail Drawing of Mold and Gate.

sunk at some point, it had too much copper oxide; but if the surface neither rose nor sunk the melt was at tough pitch, and on fracturing the sample it would present the typical tough-pitch appearance, being fine-grained to fibrous, free from blowholes, and of a rose color.

If the first sample showed a gassy melt, the cover of the crucible was removed for a minute or two, and another sample then taken. If the surface of this second sample sunk the lid was replaced for half the time that it had been previously removed. The next sample should present tough-pitch characteristics. The copper in the casting having the same composition as the molten metal in the crucible just before pouring, it

is absolutely necessary for successful work to determine the pitch accurately. If the copper is "off pitch" at the time of pouring, it is so difficult to re-treat the annealed material that it is practically useless to attempt a remelt.

The testing of the sample by the ladle, and the stirring in of the cold metallic bismuth, cools the molten copper, and this chilling is augmented by the exposure to the air. It is necessary to have absolute fluidity of the copper for a successful pouring, since the metal has to find its way through the gate and into the fingers of the mold, both of which, being of a lower temperature, tend to freeze the metal before the pour is complete.

Several melts failed on account of not preheating the mold sufficiently. A bright red heat is the proper temperature, which, however, oxidizes the graphite so rapidly that at the end of 10 or 12 melts the mold is worn out.

Rapidity of movement throughout the melt and steadiness of pouring are necessary, especially in pouring the metal into the gate. Here a slight quiver of the hand will cause the copper in the gate or fingers of the mold to separate into patches, the first patch freezing before another gets to it, and forming a "cold shut" or plane of weakness in the casting. Then, too, one disturbance begets another, and a series of such cold shuts is the result. Slowness of pouring produces a similar effect, with the added possibility of the mass freezing in the narrow neck of the gate.

With the mold in a vertical position, the air ahead of the pour had a tendency to rise through the bars and become entangled there after the copper had frozen, forming pipes sometimes a third of the length of the casting, or being distributed as blow-holes throughout the entire mass. On approaching the level, instead of being evenly distributed, the blow-holes were found more in the top of the casting, as shown at *A* in Fig. 1. An inclination of about 10° in the opposite direction removed the difficulty arising from the air contained in the mold, since it was pushed out ahead of the molten copper.

A wedge-shaped piece of graphite placed at the top of the pour-hole, like a whistle, tended to prevent any excess air from being entrained at this point, and any air so entrained was allowed to escape through a vent placed at the highest point in

the distributing passage of the gate, Fig. 1. The copper rising against the inclination of the mold gives a better chance for the air to pass up through the vent than if the mold were inclined in the opposite direction, in which latter position the molten mass would be accelerated by gravity.

Despite the care with which the specimens were cast, minute blow-holes occasionally appeared in the fractured section. In such cases an approximation of the area so occupied was made, and the stress recalculated, the final results being comparatively accurate.

Marking the Bars.

The castings were marked as follows: Each melt was given a serial number; subscript numbers 1, 2, 3, and 4 were placed at the lower right of the serial number to indicate the individual casting of the melt. The serial number is larger in size than the subscript.

Rolling the Bars.

The pieces left of the cast bars, after having been submitted to the tensile test, were heated in a gas-muffle, passed through rolls operated by a small motor, and marked with the original numbers for the purpose of making additional tensile tests. Pure copper castings were first tried, both while hot and cold, the former giving better results. Considering Hampe's remarks on the malleability of copper-bismuth alloys, I presumed that the castings would run through better cold than hot, but experiment proved otherwise. The rolled specimens were freer from checks on the edges if heated before each passage through the rolls. One annealing for the entire reduction in cross-section from the original casting to the final product of the rolls was not as good as one following each of the four steps in the reduction.

Testing the Bars.

Tensile tests were made on both cast and rolled bars, using the Riehlé testing-machine in the mechanical laboratory of Columbia University. The load was applied constantly, using the slow-speed gear; therefore no permanent set was recorded. The balance-beam fell before reaching the point of rupture, which would indicate that the specimen had stretched considerably immediately before it broke. The results of the tests, given in Tables I. and II., were obtained as follows:

If P be the breaking-force in pounds, A the area of cross-section in square inches, then p, the stress in pounds per square inch, will equal P/A. Now, if A be the original area of cross-section and a the area occupied by blow-holes, then p will equal $P/(A-a)$.

TABLE I.—*Physical Properties of Copper and Copper-Bismuth Cast Bars.*

No.	Bi.	Diameter.	Area.	Stress.	Stress.	Elongation.	Area Holes.	Corrected Area.	Corrected Stress.	Remarks.
	Per Cent.	In.	Sq. In.	Lb.	Lb. per Sq. In.	Per Cent.	Per Cent.	Sq. In.	Lb. per Sq. In.	
1_1	0	0.504	0.1995	2,200	11,020	12.5	45.0	0.1097	20,050	Turned down.
1_2	0	0.498	0.1948	2,010	10,320	09.4	55.0	0.0877	22,900	
3_1	0	0.468	0.1720	1,820	10,580	06.2	40.0	0.1032	17,600	
3_2	0	0.430	0.1452	1,910	13,150	07.1	45.0	0.0799	23,900	
6_1	0	0.546	0.2341	3,120	13,320	10.7	85.0	0.1522	20,500	Slightly columnar fracture.
6_2	0	0.548	0.2359	4,540	19,240	12.5	15.0	0.2005	22,600	
6_3	0	0.549	0.2367	4,050	17,110	20.9	20.0	0.1894	21,400	
7_1	0	0.555	0.2419	3,880	16,000	14.6	20.0	0.1935	20,000	Tough pitch.
7_2	0	0.552	0.2398	4,540	18,980	16.7	10.0	0.2154	21,100	
8_1	0	0.555	0.2419	4,490	18,560	12.5	15.0	0.2056	21,750	Not as tough as 7_1, 7_2.
8_2	0	0.560	0.2463	4,060	16,480	10.7	20.0	0.1969	20,600	
8_3	0	0.555	0.2419	2,170	8,970	4.7	20.0	0.1935	11,200	
8_4	0	0.560	0.2463	4,590	18,680	10.7	15.0	0.2093	22,000	
9_1	0.2	0.550	0.2376	5,500	23,150	12.5	15.0	0.2020	27,300	Fine-grained fracture.
9_2	0.2	0.550	0.2376	5,680	23,900	12.5	10.0	0.2188	26,500	
9_3	0.2	0.553	0.2402	6,060	25,220	12.5	05.0	0.2282	26,600	
9_4	0.2	0.555	0.2419	5,000	20,660	7.8	08.0	0.2225	22,500	
10_1	0.4	0.555	0.2419	2,500	10,330	5.6	20.0	0.1935	13,000	Columnar fracture.
11_1	0.1	0.553	0.2402	5,180	21,500	15.6	10.0	0.2161	23,900	Fine-grained fracture.
11_2	0.1	0.558	0.2445	1,890	7,780	1.8	50.0	0.1223	15,400	
11_3	0.1	0.560	0.2463	Very	coarse	grain	ed.			
12_1	0.05	0.558	0.2445	2,670	9,160	4.2	45.0	0.1345	19,800	Columnar fracture.
13_1	0.02	0.559	0.2454	3,890	15,850	5.3	35.0	0.1595	24,400	
14_1	0.01	0.563	0.2489	2,210	8,880	6.3	40.0	0.1493	14,300	
14_2	0.01	0.565	0.2507	2,100	8,370	6.3	50.0	0.1253	16,800	
15_1	0.005	0.562	0.2481	2,720	10,960	5.3	25.0	0.1861	14,600	
15_2	0.005	0.570	0.2552	3,100	12,150	7.5	30.0	0.1786	17,300	

TABLE II.—*Physical Properties of Copper and Copper-Bismuth Rolled Bars.*

No.	Dimensions.	Area.	Stress.	Stress.
	Inch.	Sq. In.	Lb.	Lb. per Sq. In.
6_2	0.257 by 0.258	0.0663	2,370	35,700
6_3	0.270 by 0.272	0.0731	2,540	34,600
7_1	0.257 by 0.258	0.0663	2,060	31,100
7_2	0.257 by 0.260	0 0668	2,550	38,300
8_1	0.256 by 0.260	0.0667	2,000	30,000
8_2	0.256 by 0.258	0.0661	1,950	29,500
8_3	0.255 by 0.255	0.0650	2,000	30,750
8_4	0.254 by 0.257	0.0650	2,000	30,750
15_1	0.254 by 0.257	0.0650	2,000	30,750
15_2	0.255 by 0.258	0.0658	1,880	28,600[a]

[a] Check at fracture.

If X be the length in inches of the casting between bearings before being pulled, and Y the length in inches stretched in that distance, then the percentage of elongation will equal Y/X.

The first castings of pure copper were turned down. The fractured section of Nos. 1_1, 1_2, Table I., showed minute blow-holes evenly distributed throughout the surface. The copper was fine-grained and presented a characteristic color. These specimens were made by the method of the Alloys Research Committee, described in their second report, but being impracticable for my special use it was abandoned. Nos. 3_1 and 3_2 were made by pouring into a vertical mold. On turning down, no imperfections in the way of blow-holes or pipes were met. After fracturing, however, several long holes were exposed, one of which was continuous for a third of the length of the casting. The occurrence of these holes indicated a mechanical disadvantage in having the molds in a vertical position. All subsequent castings were made by the method already described, with the exception of Nos. 6_1 and 6_2, which were poured in the same way but with the mold inclined in the opposite direction. The percentage of blow-holes was reduced, but the fracture was more columnar than granular. The best copper-castings produced were Nos. 7_1 and 7_2, which possess a "tough-pitch" fracture. Evidently the condition of pitch at the time of pouring these castings was just right. From the stand-point of pitch, No. 8 was very nearly as good as No. 7. With the exception of the first two melts, in which the area allowed for blow-holes was extremely large, due to the mechanical disadvantage of the method used, the other castings vary in tensile strength according to the pitch, or blow-hole content. For if the pitch be just right there will be no blow-holes present.

Discussion of Results.

Kirkaldy pulled four cast bars of copper of unstated purity, having a diameter of 0.619 in. The stress-average was 24,781 lb. per sq. in., and the elongation was 21.8 per cent. These castings had a granular fracture, which would indicate tough pitch. After allowing for blow-holes, the best results were reduced to about 22,300 lb. per sq. in. These bars were probably cast by dipping from a large mass of molten copper, in which the pitch could be more accurately determined.

No. 9, containing 0.20 per cent. of bismuth, had a tensile strength of 3,500 lb. per sq. in. more than the best copper bars. This melt differs from all other copper-bismuth alloys in that the alloy had a very fine-grained, tough-pitch fracture. Every other casting containing bismuth presented an appearance of long radiating fibers resembling pectolite. This fracture is given in Table I. as columnar, because the radiating fibers are grouped together in the form of columns. The tensile strength of castings of this columnar fracture falls much below that of pure copper and still further below that of No. 9, which was fine grained. Hampe alludes to the difference of strength of the two fractures as follows:

"All alloys of copper with metallic bismuth break easily, and show a coarse-grained, bright fracture, but if they are exceptionally fine grained they offer a much greater resistance to rupture than in the coarse-grained condition."

This difference in fracture is due probably to a difference of pitch at the time of pouring, and it is possible that bismuth itself changes the pitch.

The pieces of castings which were treated by rolling had already been strained by the tensile tests. This strain imparted the tendency to check badly on the edges in passing through the rolls. The annealing and rolling process did, however, eliminate the inaccuracy due to the presence of blow-holes in the cast bars. With one exception, all the castings of copper-bismuth either crumbled, split up the middle along a diametric plane, as did No. 9, or were so badly checked on the edges as to render them useless for the tensile tests. The exception was Nos. 15_1 and 15_2, containing 0.005 per cent. of bismuth, which fell 3,000 lb. per sq. in. below the average for pure copper. The fracture of 15_2 showed a check, which was accountable for most of the weakness. Accepting the most accurate result obtained on specimen No. 15_1, there would be but 1,800 lb. per sq. in. drop from the average for pure copper.

Conclusions.

If this difference of fracture, and hence the difference of tensile strength, of copper-bismuth alloys of the same bismuth-content is due to a difference of pitch alone, then we may consider the influence of bismuth on copper when the alloy is

fine grained. For the pitch can be so well controlled in the reverberatory furnace that a tough-pitch melt can always be obtained. Reasoning on this as a basis, an alloy of copper containing 0.18 per cent. of bismuth is stronger than pure copper, the bismuth here presenting the same effect as do arsenic and antimony, its associates in the Periodic Law.

For copper to be rolled, the allowable percentage of bismuth is governed by the limiting quantity which can be present without appreciably lowering the malleability and ductility of the alloy. This limit was found to be less than 0.005 per cent. of bismuth for metal to be rolled, either when hot or cold. If this limit be exceeded, the ductility of the copper is so lowered as to interfere with the process of wire-drawing. It is reasonable to assume that the presence of so small an amount of bismuth will not appreciably lessen the electric conductivity of the copper.

Hampe's limits of the percentage of bismuth which will cause hot- and cold-shortness of the alloy seem to be just the reverse. So that 0.02 per cent. of bismuth makes copper cold-short and 0.05 per cent. hot-short.

Acknowledgment.

My sincere thanks are due to Dr. Myrick N. Bolles, of the Department of Metallurgy, Columbia University, for his guidance and valuable assistance in the work of this thesis; also to Mr. Thompson, of the same department, for making the quantitative determination for bismuth, which served as a check on the synthetic analysis of the alloy. For assistance in conducting the tests on the Riehlé machine, I am indebted to Prof. I. H. Woolson and his assistants in the mechanical laboratory of Columbia University.

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Bulletin of the American Institute of Mining Engineers.

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SECTION I.—INSTITUTE ANNOUNCEMENTS.

This section contains announcements of general interest to the members of the Institute, but not always of sufficient permanent value to warrant republication in the volumes of the *Transactions*.

SECTION II.—TECHNICAL PAPERS AND DISCUSSIONS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

A detailed list of the papers contained in this section is given in the Table of Contents. They have been so printed and arranged (blank pages being left when necessary) that they can be separately removed for classified filing, or other independent use.

A small stock of separate pamphlets, duplicating the technical papers given in Section II. of this Bulletin, is reserved for those who desire extra copies of any single paper.

Comments or criticisms upon all papers given in this section, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

All communications concerning the contents of this Bulletin should be addressed to Dr. Joseph Struthers, Assistant Secretary and Editor, 29 W. 39th St., New York, N. Y. (Telephone number 4600 Bryant).

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

INSTITUTE ANNOUNCEMENTS.

Spokane Meeting and Excursions.

Advices from the party participating in the tour through Yellowstone Park and the excursions to Butte and Anaconda, preceding the Institute meetings at Spokane, are to the effect that the trip is one of the most successful of the many delightful excursions held by the Institute. A detailed account of the meeting and excursions will be printed in a later number of the *Bulletin.*

Meetings of Other Societies.

American Society of Mechanical Engineers.—The November meeting of the American Society of Mechanical Engineers will be held in the Engineering Societies Building, New York, on the evening of November 9. The annual meeting will be held in New York December 7–10. Meetings of the Society are also to be held throughout the fall and winter in St. Louis and Boston.

In connection with the Hudson-Fulton celebration in September and October in New York, and as a contribution on the part of the engineering profession, there has been placed in the rooms of the American Society of Mechanical Engineers a valuable exhibit of objects of interest relating to the early history of steam-navigation. The exhibit includes the following objects belonging to the Society: an oil-portrait of Robert Fulton by himself; original drawings by Fulton; a table that belonged to Fulton; reproduction in bronze of the Fulton drawing of the steamer *Potomac*, 1820; letters of Fulton's workmen concerning the first steamboat; portraits of prominent marine engineers, etc. Included in the exhibit are a collection of Ericsson models from the United Engineering Society, and models of early steamboats, including Fulton's *Clermont*, from the Smithsonian Institution.

American Institute of Electrical Engineers.—The November meeting of the American Institute of Electrical Engineers will be held in the auditorium of the Engineering Societies Building, 33 West Thirty-ninth St., New York, on Friday evening, Nov. 12, 1909, at 8 o'clock. Dr. Cary T. Hutchinson, consulting engineer, of New York, will present a paper entitled: The Electric System of the Great Northern Railway Company at Cascade Tunnel.

Members of the American Institute of Mining Engineers are cordially invited to attend these meetings.

Conservation of Mineral Resources.

The papers on the Conservation of Mineral Resources, by members of the U. S. Geological Survey, which were included in the Report of the National Conservation Commission, have been reprinted in *Bulletin No.* 394 *of the U. S. Geological Survey*, entitled, *Papers on the Conservation of Mineral Resources.* These papers are:

Coal-Fields of the United States. By M. R. Campbell and E. W. Parker.
Estimates of Future Coal Production. By Henry Gannett.
The Petroleum Resources of the United States. By D. T. Day.
Natural-Gas Resources of the United States. By D. T. Day.
Peat Resources of the United States, Exclusive of Alaska. By C. A. Davis.
Iron-Ores of the United States. By C. W. Hayes.
Resources of the United States in Gold, Silver, Copper, Lead, and Zinc. By Waldemar Lindgren.
The Phosphate Deposits of the United States. By F. B. Van Horn.
Mineral Resources of Alaska. By A. H. Brooks.

A large edition of this *Bulletin* has been printed, and copies may be obtained without charge on application to the Director of the U. S. Geological Survey, Washington, D. C.

Office Facilities for Visiting Members.

A separate room in the suite occupied by the American Institute of Mining Engineers on the ninth floor of the United Engineering Society Building, has been equipped with furniture and telephone extension for the temporary use of mem-

bers of the Institute or of sister societies, or visitors suitably accredited.

Members of the Institute visiting New York for a short time, who need office facilities during their stay, or members residing in the city who need temporary office accommodation, can arrange to have set apart for their exclusive use a room, equipped with office furniture, telephone, etc., in the suite of the Institute. It is not the intention to give possession of the room to any individual for an indefinite time, but to offer to members of the Institute an opportunity to acquire a well-located, well-equipped business headquarters to carry on transactions which would not warrant the establishment of a permanent office. The room devoted to this purpose is entirely separate from the reception- and writing-rooms for the general use of the members. A small fee will be required for the use of the facilities furnished. For the conditions of this privilege, inquiry should be made at the office of the Secretary of the Institute.

How to Use the "Transactions" of the Institute.

Buy a copy of the Complete Analytical and Alphabetical Index of Volumes I. to XXXV., inclusive.

If you own a full set of the *Transactions*, this Index will make the whole of it instantaneously available without detailed research into each volume separately.

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In short, to those who own complete sets of the *Transactions*, this Index will be a great convenience; but to those who do not, it will be a professional necessity.

This volume is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub headings, and including

abundant cross-references. It has not been stereotyped, and the edition is limited to 1,600 copies. The price of the volume, bound in cloth, is $5, and bound in half-morocco to match the *Transactions*, $6. The delivery charges will be paid by the Institute on receipt of the above price.

Hydrographic Chart.

Owing to the great value to hydrographers of the chart contained in the paper, A Graphic Solution of Kutter's Formula, by L. I. Hewes and Joseph W. Roe (*Bulletin No.* 29, May, 1909, p. 454), a special edition for office or field use has been printed on durable cloth. Copies of this separate chart may be obtained, at a cost of 50 cents each, on application to the office of the Secretary of the Institute.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

RULES.

For the protection and convenience of members, the following rules have been adopted:

The Secretary of each Society will, upon application, issue to any member of his Society in good standing a personal, non-transferable card, entitling him to the use of the Libraries in the alcoves of the Reading-Room.

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The librarians are not permitted to lend to any person any catalogued pamphlet or volume, unless authorized in writing so to do by the Secretary or Chairman of the Library Committee of the Society to which the pamphlet or volume belongs.

Any person discovering a mutilation or defect in any book of the libraries is requested to report it to the librarian on duty.

Library Additions.

From Sept. 1 to Oct. 1, 1909.

AMERICAN CERAMIC SOCIETY. Transactions. Vol. XI. Columbus, 1909. (Gift.)

AMERICAN ELECTROCHEMICAL SOCIETY. Transactions. Vol. 15. South Bethlehem, 1909. (Exchange.)

AMERICAN MINING CONGRESS. First Report of Committee on Prevention of Accidents in Metalliferous Mines, Dec., 1908. N. p., n. d. (Gift.)

BLACK DIAMOND'S DIRECTORY OF COAL-MINE OPERATORS, 1909. Chicago, 1909. (Purchase.)

BROKEN HILL SOUTH SILVER MINING CO. Reports, Statements of Account, etc., for Half-Year ended June 30, 1909. Melbourne, 1909. (Gift of Mr. Wainwright.)

CATALOGUE OFFICIEL DE LA SECTION ALLEMANDE. (Exposition Universelle de 1900.) Berlin, 1900. (Gift of Prof. Kunz.)

CENTURY OF POPULATION GROWTH FROM THE FIRST CENSUS OF THE UNITED STATES TO THE TWELFTH, 1790–1900. Washington, U. S. Government, 1909. (Exchange.)

CHART OF THE INDIANA COAL-FIELD, 1909. (Exchange.)

COLLOID MATTER OF CLAY AND ITS MEASUREMENT. (Bulletin No. 388, U. S. Geological Survey.) By H. E. Ashley. Washington, U. S. Government, 1909. (Exchange.)

DRY FARMING IN WYOMING. By V. T. Cooke. Cheyenne, 1909. (Exchange.)

FORDHAM UNIVERSITY. School of Law Announcement, 1909–1910. New York, 1909. (Gift.)

GEOLOGICAL MAP OF OHIO, 1909. (Exchange.)

GEOLOGY AND ORE-DEPOSITS OF GOLDFIELD, NEVADA. (Professional Paper No. 66, U. S. Geological Survey.) By F. L. Ransome. Washington, U. S. Government, 1909. (Exchange.)

INDIA—MINES DEPARTMENT. Report of the Chief Inspector of Mines, 1908. Calcutta, 1909. (Exchange.)

INDIANA GEOLOGY AND NATURAL RESOURCES DEPARTMENT. Annual Report, 33d. Indianapolis, 1909. (Exchange.)

INSTITUTION OF MINING AND METALLURGY. Transactions. Vol. XVIII. London, 1909. (Exchange.)

JURISDICTION AND WORK OF THE PUBLIC SERVICE COMMISSION, FIRST DISTRICT. N. p., n. d. (Gift.)

KATALOG DER BIBLIOTHEK DES KAISERLICHEN PATENTAMTES. 8 Nachtrag. Berlin, 1909. (Exchange.)

KONIGLICHE SÄCHSISCHE BERGAKADEMIE ZU FREIBERG. Programm, 1909–1910. Freiberg, 1909. (Exchange.)

MARRIAGE AND DIVORCE, 1867–1906. Pt. 1. (U. S. Census Bureau. Special Reports.) Washington, U. S. Government, 1909. (Exchange.)

MINERAL CONTENT OF ILLINOIS WATERS. (Bulletin No. 10, Illinois Geological Survey.) Urbana, 1909. (Exchange.)

MINERAL RESOURCES OF WYOMING. Cheyenne, n. d. (Exchange.)

MYSORE GEOLOGICAL DEPARTMENT. Report of the Chief Inspector of Mines, 1907–08. Bangalore, 1909. (Exchange.)

NATIONAL CONFERENCE ON STANDARD ELECTRICAL RULES. Minutes of 7th Annual Meeting held at the Engineering Building, Mar. 26, 1909. (Purchase.)

NEWARK (N. J.) BOARD OF STREET AND WATER COMMISSIONERS. Annual Report, 1908. Newark, 1908. (Gift.)

NEW JERSEY GEOLOGICAL SURVEY. Annual Report of the State Geologist, 1908. Trenton, 1909. (Exchange.)

NEW YORK (STATE) ADVISORY BOARD OF CONSULTING ENGINEERS. Report to the Governor upon its Work relating to the Barge Canal from Jan. 1, 1908, to Jan. 1, 1909. Albany, 1909. (Gift.)

NOUVEAU LAROUSSE ILLUSTRÉ AND SUPPLEMENT. 8 vols. Paris, n. d. (Purchase.)

OFFICIAL HANDBOOK OF THE CEYLON COURT. (St. Louis World's Fair, 1904.) Colombo, 1904. (Gift of Prof. Kunz.)

ONE HUNDRED YEARS OF NEW YORK STATE GEOLOGIC MAPS, 1809–1909. By Henry Leighton. Albany, 1909. (Gift of Author.)

PAPERS ON THE CONSERVATION OF MINERAL RESOURCES. (Bulletin No. 394, U. S. Geological Survey.) Washington, U. S. Government, 1909. (Exchange.)

PAPERS ON THE CONSERVATION OF WATER RESOURCES. (Water-Supply Paper No. 234, U. S. Geological Survey.) Washington, U. S. Government, 1909. (Exchange.)

PENNSYLVANIA—AGRICULTURAL DEPARTMENT. Bulletin Nos. 172–74, 1877–78. Harrisburg, 1908–09. (Exchange.)

——— ——— DAIRY AND FOOD DIVISION. Monthly Bulletin. Vol. 6, No. 12; Vol. 7, Nos. 2–5. Harrisburg, 1909. (Exchange.)

——— ——— Bulletin No. 183. Harrisburg, 1909. (Exchange.)

——— ——— ZOOLOGY DIVISION. Bulletin. Vol. 6, Nos. 9–12. Harrisburg, 1909. (Exchange.)

——— ——— BUREAU OF CHEMISTRY. Bulletin Nos. 175–76. Harrisburg, 1909. (Exchange.)

——— AUDITOR GENERAL. Report on the Finances, 1908. Harrisburg, 1909. (Exchange.)

——— BANKING COMMISSIONER. Annual Report, 14th, 1908, Pt. 1. Harrisburg, 1909. (Exchange.)

——— ——— Recommendations by . . . as made to Governor E. S. Stuart, Nov., 1908. Harrisburg, 1909. (Exchange.)

——— FACTORY INSPECTOR. Annual Report, 1908. Harrisburg, 1909. (Exchange.)

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——— GENERAL ASSEMBLY. Laws of Pennsylvania, 1909. Harrisburg, 1909. (Exchange.)

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——— Health Bulletin No. 1. July, 1909. (Exchange.)

——— HEALTH COMMISSIONER. Annual Report, 2d, 1907. Harrisburg, 1908. (Exchange.)

——— INTERNAL AFFAIRS DEPARTMENT. Annual Report: Pt. 1—Land Office, Boundary Lines, 1908. Pt. 2—Assessments, Taxes, 1908. Pt. 4—Railroads, Canals, etc., 1908. Harrisburg, 1908–1909. (Exchange.)

——— PUBLIC INSTRUCTION DEPARTMENT. Manual for Borough and Township High Schools of Pennsylvania, 1909. Harrisburg, 1909. (Exchange.)

——— ——— Proceedings of the Annual Meeting of the Board of Principals of the State Normal Schools of Pennsylvania, 1908. Harrisburg, 1909. (Exchange.)

PENNSYLVANIA—PUBLIC INSTRUCTION DEPARTMENT. List of High Schools in the Several Counties of Pennsylvania, 1908. Harrisburg, 1909. (Exchange.)

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——— SOLDIERS' AND SAILORS' HOME AT ERIE. Report of Board of Trustees, 1906–08. Harrisburg, 1909. (Exchange.)

——— STATE BOARD OF AGRICULTURE. Program of the Spring Meeting of the Farmers' Annual Normal Institute, May 25–28, 1909. Harrisburg, 1909. (Exchange.)

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DENVER ENGINEERING WORKS Co., Denver, Colo.

Reprints of drawings, made in the drafting-rooms of the company, and representing:

a, Perspective view of Colorado School of Mines Ore-Dressing Plant.

b, Plan of Colorado School of Mines Ore-Dressing Plant.

c, Sectional elevation, GG, through Colorado School of Mines Ore-Dressing Plant.

d, Sectional elevation, DD, through Colorado School of Mines Ore-Dressing Plant.

ECONOMY DRAWING TABLE Co., Toledo, Ohio. Drawing-Tables, Sectional Filing-Cases, and specials in this line for engineers, architects, contractors, technical schools, etc.

EIMER & AMEND, 211 Third Ave., New York, N. Y. Catalogue of Assay and Metallurgical Laboratory Supplies, as balances, burners, crushers, tank-outfits, gas-generators, sampling-apparatus, etc.

ELECTROCHEMICAL AND METALLURGICAL INDUSTRY, New York, N. Y. Dictionary of Chemical and Metallurgical Material. An alphabetically-arranged dictionary of chemical and metallurgical machinery, appliances, and material, giving concise and accurate descriptions of the different makes of apparatus, and lists of firms handling them.

FAIRBANKS Co., New York, N. Y.

Bulletins on:

Hyatt Flexible Roller-Bearings, reducing friction to a minimum, and increasing the efficiency of machinery.

Oneida Steel Pulleys for transmission.

GENERAL ELECTRIC Co., Schenectady, N. Y.

Folders on:

Measuring-Instruments, as wattmeters, ammeters, voltmeters, etc.

Fan-Motors for residences and telephone-booths.

Edison Gem Lamps for standard multiple lighting-circuits, 100 to 130 volts.

The new 40-Watt G. E. Tungsten Lamp, a lamp giving good light with small current.

Tungsten Lamp Installation in a New York Office Building, resulting in a saving of current and an increase of volume of light.

Tungsten Lamp Logic; showing how the lamp is applicable to the home, the office, and the factory, and the small amount of current it consumes in contrast to other types.

Bulletin No. 4675, June, 1909. Single-Phase Motors, Type RI, giving illustrations and description of construction.

Bulletin No. 4680, July, 1909. Sign-Lighting with G. E. Tungsten Lamps, with data, showing saving in the use of this type of lamps for illuminated signs.

GENERAL ELECTRIC Co., Schenectady, N. Y.

Bulletin No. 4687, Aug., 1909. Direct-Current Motor-Starting Rheostats, CR-107 and CR-111, with illustrations and descriptive paragraphs of construction.

Bulletin No. 4688, Aug., 1909. Compound Meter-Board, made of insulated compound, non-inflammable, non-warping, and more expedient than wooden meter-board.

HENDRYX CYANIDE MACHINERY Co., Denver, Colo. Catalogue of Cyaniding Machinery, with illustrations, diagrams, and articles on testing, method of operation, Hendryx combination agitator and filter, tailings-dewaterer, melting-furnaces, and precipitation-presses.

J. GEO. LEYNER ENGINEERING WORKS Co., Littleton, Colo.

Leyner Bulletin No. 1008, July 20, 1909. Drill-Sharpeners Nos. 1, 2, and 3, with cuts and articles on their advantages, construction, and instruction to operators.

Bulletin No. 1007, July 10, 1909. Leyner Hand Drills Nos. 1 and 2, showing construction and giving explanatory paragraphs on same.

ONEIDA STEEL PULLEY Co., Oneida, N. Y. Folders on Oneida Steel Pulleys, with cuts of pulleys, tables of sizes, price-list of split-pulleys, and notes on their advantages and economy.

WASHBURN & GRANGER, New York, N. Y. Catalogue "B" of "Dean" Grates, dumping, shaking, and stationary makes, built for furnaces using either pea coal, rice coal, buckwheat coal, or bituminous coal.

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MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members or associates who accepted election during the month of September, 1909:

Members.

Alfred P. Busey, Jr.,	Campo Seco, Cal.
Edwin J. Collins,	Duluth, Minn.
John R. Finletter,	Globe, Ariz.
L. H. French,	New Rochelle, N. Y.
John Greenall,	Allentown, Pa.
Herbert E. Jackman,	Cobalt, Ontario, Can.
H. Seaver Jones,	Oxford, N. J.
Howard W. Kitson,	Colorado Springs, Colo.
Seth S. Langley,	Indé, Durango, Mex.
Percy A. Wagner,	Johannesburg, Transvaal, S. Africa.
George H. Warren,	Minneapolis, Minn.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed for election as members of the Institute during the month of September, 1909. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

John Alexander Agnew,	Kalgoorlie, Western Australia.
Donald James Browne,	Rossland, B. C., Can.
Archibald Felee Dick-Cleland,	Guadalajara, Jalisco, Mex.
Raoul Geer Dufourcq,	Arizpe, Sonora, Mex.
Nathaniel Grant,	Poland, Ariz.
James Harold Hance,	Salt Lake City, Utah.

Archibald Stewart Hummell, ... High Bridge, N. J.
Percy Kenyon, ... Conception del Oro, Zacatecas, Mex.
Eugene McAuliffe, ... Chicago, Ill.
H. A. Morin, ... Gow Ganda, Ontario, Can.
James Wilfrid Newbery, ... Kalgoorlie, Western Australia.
Benjamin Gilmore Patterson, ... Mount Morgan, Queensland, Australia.
William Paterson Rutherford, Jr., ... Batoum, South Russia.
Sydney Latham Shonts, ... Wallace, Idaho.
Tsok Kai Tse, ... Wickenburg, Ariz.
Leslie James Wilmoth, ... Germiston, Transvaal, S. Africa.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of September, 1909. This list, together with the lists given in the *Bulletin*, Nos. 26 to 33, for February to September, therefore, supplements the annual list of members corrected to Jan. 1, 1909, and brings it up to the date of Oct. 1, 1909. The names of Members who have accepted election during the month (new members), are printed in *italics*.

ABADIE, EMILE R. .... 703 Security Bldg., Los Angeles, Cal.
ALLEN, JOHN F., Cons. Engr. .... 62 London Wall, London, E. C., England.
ALLEN, ROBERT, Care Cons. Goldfields of S. A., Ltd., P. O. Box 1167, Johannesburg, Transvaal, So. Africa.
BAILEY, J. TROWBRIDGE .... 32 W. 40th St., New York, N. Y.
BATES, MOWRY .... Care Hotel Cobalt, Cobalt, Ont., Canada.
BATZ, BARON RENE DE .... 2 Ave. Camoëns, Paris, France.
BEATTY, A. CHESTER, Cons. Engr. .... 71 Broadway, New York, N. Y.
BECK, EDWIN L. .... Jardine, Mont.
BERTOLET, ALFRED S. .... 10618 Torrence Ave., Chicago, Ill.
BILLIN, CHARLES E. .... 2632 Lake View Ave., Chicago, Ill.
BISHOP, ROY N. .... Room 1121, 1st National Bank Bldg., San Francisco, Cal.
BODDINGTON, HENRY D., Mineral de Nueva Union, Distrito de Rayõn, Chih., Mexico.
BOTSFORD, ROBERT S., Mine Mgr., Brazilian Development Synd., Ltd., Làvras, Rio Grande do Sul, Brazil, So. Amer.
BRADLEY, DUDLEY H., JR. .... Care Shannon Copper Co., Metcalf, Ariz.
BRADLEY, PHILIP R., Exploration Co. of New York, Room 1, 15 Broad St., New York, N. Y.
BRIGGS, WILLIAM A. J. .... Renison Bell Tin Mine, Zeehan, Tasmania.
BROOK, REGINALD H. T., Genl. Mgr., New Wyengatta & West Wyengatta G. M. Co., Golconda, via Launceston, Tasmania.
BURLS, HERBERT T., Care Edward Riley & Harbord, 16 Victoria St., Westminster, S. W., London, England.
**Busey, Alfred P., Jr.*, Min. Engr., Mgr. Penn Chemical Wks., Campo Seco, Cal. '09.
CARPENTER, FRANK R. .... 1065 Pearl St., Denver, Colo.

CHALMERS, THOMAS S., Care Chalmers & Williams, Commercial National Bank Bldg., Chicago, Ill.
CHANNING, J. PARKE....................................42 Broadway, New York, N. Y.
CHURCH, ALBERT K..Hampton, N. H.
CLARK, HORACE H..................................208 So. 43d Ave., Chicago, Ill.
*_Collins, Edwin J._, Cons. Min. Engr....1008-09 Torrey Bldg., Duluth, Minn. '09.
COLLINS, FRANCIS W., Care Construction Service Co., 15 William St., New York, N. Y.
COLLINS, GLENVILLE A....................................P. O. Box 1845, Seattle, Wash.
COLVOCORESSES, GEORGE M., Supt., Millerett Silver Mining Co., Gowyanda, Ont., Canada.
COULDREY, PAUL S., Care British Columbia Copper Co., Ltd., Greenwood, B. C., Canada.
COURTIS, WILLIAM M., Min. Engr. and Met......449 Fourth Ave., Detroit, Mich.
CRAWFORD, WALTER H., Cons. Min. Engr., 181 Huntington Ave., Boston, Mass.
DARBY, THOMAS L...Bluff, Utah.
DARGIN, PERCY W....................................Box 450, Sta. C, Los Angeles, Cal.
DAVELER, ERLE V., Min. Engr.......Utah Copper Mill, Bingham Canyon, Utah.
DAVIS, FLOYD, Cons. Min. Engr. and Met............1659 Broadway, Denver, Colo.
DAVIS, F. HARLEY, Prest., Davis Drill Co., Brown-Marx Bldg., Birmingham, Ala.
DE LASHMUTT, IVAN...................Highland Boy Mine, Bingham Canyon, Utah.
DENNIS, WILLIAM B., Genl. Mgr., Black Butte Quicksilver Co., Black Butte, Ore.
DISSINGER, EARL, Min. Engr..................Quinn River Crossing, via Amos, Nev.
DIXON, ALEXANDER G., Ferreria de Encarnacion, Estado de Hidalgo, Dist. Zimapan, Mexico.
DOMINIAN, LEON...........................Room 914, 29 Broadway, New York, N. Y.
DONNELLY, THOMAS F...............................Davis Pyrites Mine, Davis, Mass.
DOWD, JOHN H.......................... 9 Rose Mount, Wallington, Surrey, England.
DRAPER, JAMES C., Min. Engr........................Miners' Bank Bldg., Joplin, Mo.
*_Edwards, Henry W._, Met.........810 So. Pennsylvania Ave., Denver, Colo. '93.
ELMER, WILLIAM W., Cons. Min. Eng.....Coneto, via Guatimape, Dur., Mexico.
*_Finletter, John R._, Mining..................Care Hotel Dominion, Globe, Ariz. '09.
FLYNN, FRANCIS N., Min. Engr. and Met......................................Globe, Ariz.
*_French, L. H._, Mining and Railroad.......................New Rochelle, N. Y. '09.
FROEHLING, HENRY.....................................911 Floyd Ave., Richmond, Va.
GOODALE, STEPHEN L., Asst. Prof. of Met., Univ. of Pittsburg, School of Mines Bldg., Grant Blvd., Pittsburg, Pa.
*_Greenall, John_, Engr..........................1503 Hamilton St., Allentown, Pa. '09.
HALL, GEORGE.................."Steephill," Boxwell Road, Berkhamstead, England.
HAMMAN, JOHN S...P. O. Box 654, Boise, Idaho.
HANCOCK, STRANGMAN.....................Kennell Holt, Cranbrook, Kent, England.
HAWXHURST, ROBERT, JR........................2907 Fillmore St., San Francisco, Cal.
HENDERSON, HARRY P...Goldfield, Nev.
HEWETT, D. FOSTER...............................199 Mansfield St., New Haven, Conn.
*_Jackman, Herbert E._, Min. Engr......P. O. Box 550, Cobalt, Ont., Canada. '09.
*_Jones, H. Seaver_, Blast Furn. Supt., Empire Steel & Iron Co., Oxford, N. J. '09.
JOSEPHS, IRVING S.....................................300 W. 109th St., New York, N. Y.
KAEDING, GEORGE L..........Kaeding Construction Co., 556 Vine St., Reno, Nev.
KAMIYAMA, TATSUZO....................No. 202 Harashuki, Aoyama, Tokyo, Japan.
KENT, JOSEPH F., Genl. Mgr., Stone Branch Coal Co., Stone Branch, Logan Co., W. Va.

KING, HAROLD F...... ....61 Maiden Lane, Kingston, N. Y.
**Kitson, Howard W.*, Min. Engr., 330½ S. Tejon St., Colorado Springs, Colo. '09.
KRUTTSCHNITT, JULIUS, JR., Supt. Reforma Unit., La Reforma, C. Cienegas, Coah., Mexico.
LAIRD, GEORGE A., Mgr., Candelaria Min. Co., Care Smith & Laird, Bisbee, Ariz.
LAMB, ROBERT B.........Care C. L. Constant Co., 42 Broadway, New York, N. Y.
**Langley, Seth S.*, Min. Engr...Indé Gold Mining Co., Indé, Dur., Mexico. '09.
MCNULTY, SAMUEL G..Rose Siding, W. Va.
MACDONALD, A. C., Care Macdonald, Gibbs & Co., 3 and 4 Great Winchester St., London, E. C., England.
MACNUTT, C. H., Genl. Mgr., Poderosa Mining Co., Ltd., Antofagasta, Chile, So. Amer.
MANNING, JOHN F., Min. Engr., Seoul Mining Co., Care Dr. J. H. Wells, Pyeng Yang, Korea.
MATTHIESSEN, F. W................ ....Hot Springs, Ark.
MOUBRAY, JOHN M., Min. Engr......98 Leadenhall St., London, E. C., England.
MUSGRAVE, ROBERT...Santa Cruz, Cal.
O'BRIEN, THOMAS H., Genl. Mgr...............Stag Cañon Fuel Co., Dawson, N. M.
REDFEARN, ALBERT M., Mill Foreman for Vanadium Alloys Co., Newmire, San Miguel Co., Colo.
RITTER, ETIENNE A.................1712 No. Nevada Ave., Colorado Springs, Colo.
RHODES, F. B. F...................Care Am. Smelting & Refining Co., Maurer, N. J.
ROSE, HUGH...2a Calle Berlin 15, Mexico City, Mexico.
SCHWARZ, T. E...106 Marion St., Brookline, Mass.
SHAW, RICHARD C., Min. Engr........................68 Post St., San Francisco, Cal.
SHEPARD, FRANK E................................. ......P. O. Box 1802, Denver, Colo.
SHORT, FRANK R............. ....Carson City, Nev.
SIMON, TREVOR B................ ....124 King Ave., Columbus, Ohio.
SMITH, R. NEIL...........................Melbourne Club, Melbourne, Vic., Australia.
TAFT, HARRY H...................... ....Hotchkiss, Colo.
THURSTON, E. COPPEE.................................80 Maple Ave., Morristown, N. J.
TURNER, C. EDWARD, Huelva Copper and Sulphur Mines, Valdelamusa, Huelva, Spain.
TURNER, SCOTT...326 Franklin Ave., Lansing, Mich.
VIVIAN, JOSEPH C., Underground Mgr., Gt. Boulder Perseverance Gold Mine, Finiston, West Australia.
**Wagner, Percy A.*, Acting Prof. of Geol., Transvaal Univ. College, Johannesburg, Transvaal, So. Africa. '09.
WALTER, E. W.......................................Care Louis Wyman, Silverton, Colo.
***Warren, George H.*, Explorer and Developer of Mineral Lands, 3443 So. Irving Ave., Minneapolis, Minn. '09.
WEBSTER, ERASTUS H., Mgr., Cia. Minera La Republica, S. A., Ocampo, Dist. Rayõn, Chih, Mexico.
WILLIAMS, EDWARD G...............................43 Exchange Pl., New York, N. Y.
WILLIAMS, RALPH B..Ray, Ariz.
WOLF, HARRY J., Min. and Cons. Engr., Western Metals Co., and Stanley Mines Co., 1748 Welton St., Denver, Colo.
WOLFF, MARK A., Care Inst. of Min. and Met., Salisbury Ho., London, E. C., England.
WORTH, JOHN G...621 So. Virginia St., Reno, Nev.
WRIGHT, CHARLES W.................Societá di Pertusola, Inglesias, Sardinia, Italy.
YOUNG, GEORGE J......Care Mackay School of Mines, Univ. of Nev., Reno, Nev.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address on Records, from which Mail has been Returned.
Adams, Randolph,	Copperhill, Tenn.
Alexander, George E.,	Sparta, Ore.
Allen, Frederick E.,	Bloomsburg, Pa.
Andrew, Thomas,	Pretoria, So. Africa.
Arozarena, R. M. de,	Mexico City, Mexico.
Bartoccini, Astolfo,	214 E. 90th St., New York, N. Y.
Bassett, Thomas B.,	Cumpas, Sonora, Mexico.
Batchelder, Joseph F.,	54 1st St., Portland, Ore.
Bellam, Henry L.,	Reno, Nev.
Bouchelle, James F.,	22 Duncan Ave., Jersey City, N. J.
Brook, Henry E. C.,	Cadia, N. S. W., Australia.
Brown, Frank H.,	Coppermount, Alaska.
Campa, Jose,	Mexico City, Mexico.
Cragoe, A. Spencer,	Vencedora, Mexico.
Derby, Harry S.,	134 Monroe St., Chicago, Ill.
Dickson, George H.,	Lethbridge, Alberta, Canada.
Dougherty, Clarence E.,	41 Wall St., New York, N. Y.
Ekberg, Benjamin P.,	Johannesburg, Transvaal, So. Africa.
Field, Wilfrid B.,	Mexico City, Mexico.
Fitzsimmons, F. J.,	Cananea, Mexico.
Francis, George G.,	177 St. George's Sq., London, W., England.
Fuller, Frederick D.,	Sumpter, Ore.
Gage, Edward C.,	San Dimas, Dur., Mexico.
Gee, Emerson,	Reno, Nev.
Hawkins, Tancred,	Ballydehob, Ireland.
Hunt, Thatcher R.,	Iron Mt., via Keswick, Cal.
Jackson, Byron N.,	Milton, Cal.
Jessop, Herbert J.,	Guanacevi, Mexico.
Jewett, Eliot C.,	2918 Morgan St., St. Louis, Mo.
Judd, Henry A.,	Mertondale, W. Australia
King, Rufus H.,	Union Club, New York, N. Y.
Kow, Tong Sing,	Shanghai, China.
Mildon, Reginald B.,	Nacozari, Son., Mexico.
Moulton, Herbert G.,	Cobalt, Ont., Can.
Muir, Thomas K.,	Portland, Ore.
Nawatny, William F.,	Harrisburg, Ill.
O'Byrne, Joseph F.,	Midas, Nev.
Philbrick, Arthur,	Manhattan, Nev.
Piper, John W. H.,	Buenos Ayres, Argentine Rep., S. A.
Potter, J. A.,	41 W. 124th St., New York, N. Y.
Rigney, Thomas P.,	Reno, Nev.
Rodda, Richard W.,	Seattle, Wash.
Sandifer, Harmer C.,	El Oro, Mexico.
Schlemm, William H.,	Durango, Mexico.
Scott, Winfield G.,	Long Beach, Cal.
Skelding, Joseph F.,	Embreeville, Tenn.
Thomas, Richard A.,	43 Wall St., New York, N. Y.
Vaux, Charles A.,	P. O. Box 80, East Rand, So. Africa.
Vidler, Louis W.,	Lookout Mountain, Colo.

Warren, Henry L. J.,	Salt Lake City, Utah.
Wiswell, Herbert J.,	Carterville, Mo.
Wolfe, Burton L.,	Ely, Nev.
Young, William,	Kenora, Ont., Canada.

NECROLOGY.

The deaths of the following members have been reported to the Secretary's office during September, 1909:

Date of Election.	Name.	Date of Decease.
1908.	*John M. Grice,	September 23, 1909.
1903.	*Harold H. Harvey,	August 24, 1909.

* Member.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Borax-Deposits of the United States.

BY CHARLES R. KEYES, DES MOINES, IOWA.

(Spokane Meeting, September, 1909.)

CONTENTS.

I. INTRODUCTION.

A COMPLETE transformation has taken place in the borax industry during the year 1908. A most remarkable factor in

this radical change in method of producing the crude borates has been its removal from the realm of industrial chemistry to the field of mining. With the development of extensive deposits of borate-minerals interstratified in thick sequences of Tertiary clays and sands, their winning becomes a strictly mining-enterprise, of the same kind and of the same certainty as the digging of coal or iron-ore.

The major supply of the world's production of commercial borax now comes from the United States. While formerly all of the boric salts of commerce were laboriously extracted from the waters of saline lakes of the arid regions, or from the bottom-salts of desiccated ponds, it later was largely slowly leached from ancient desert shales and clays. During these periods the borax industry was a very hazardous and expensive vocation.

The discovery of large deposits of very pure, crystallized borate-minerals in the old Tertiary clays of southern California has enabled the main borax-supplies of the United States to be drawn from a single locality in the Mojave desert, near Daggett. The later finding of immense deposits of the crystallized mineral within easy access to good transportation-facilities promises not only to alter the character of the borax industry for the entire world, but to reduce the cost of production to less than one-third of the present figures.

Some of the extensive bedded deposits of borate-minerals recently discovered and investigated are located in a country that has been always one of the most inaccessible places of our domain. The geology of the region has been wholly unknown. While borates had been recorded from the district, the importance of the deposits has never been determined. The geographic extent of even the shales carrying the borates, or likely to contain them, has only been suggested in the vaguest manner, and then with no association of commercial values. The stratigraphy of the borax-minerals is, therefore, at the present time, of exceptional interest. In the Death Valley district the Tertiary clay-beds, of great extent and thickness, are perhaps as finely displayed as anywhere else. Many of the geological facts associated with the occurrence of the borax are also worthy of more than passing notice.

The present account of the geology of the borax-deposits in

the United States originated incidentally in a commercial inquiry regarding the future of certain kinds of borax-supplies, undertaken for one of the chief borax-producing companies.

Later was begun a more personal inquiry of the more strictly geologic features of the occurrence of borate-minerals generally. Extensive investigation into metalliferous mining-properties in the neighborhood of certain of the larger borate-deposits gave opportunity to work out more in detail the geology of the districts in a way which before it was impossible satisfactorily to do. The results of these observations in the Death valley, the Mojave desert, and the Santa Clara valley are given herewith. The more strictly industrial aspects and treatment of the borax-substances for the market will be described in a later paper, after the deposits of other parts of the world have been inspected, and especially those of South America, Turkey, Italy, Germany, and India.

II. Occurrence.

It is now nearly 50 years since borax was first produced in commercial quantities in the United States. Since that time, about 1864, the industry has undergone several distinct changes, and is now entering upon its fourth important stage.

During the early period, soon after the discovery of the presence of boric acid in the waters of Clear lake, in northern California, lake-waters were evaporated and the boric salts extracted from the residues. This method prevailed for the period from 1864 to 1872.

In 1874 it was found that the crusts formed on the surface of certain desert marshes were rich in boric contents. From 1872 to 1890 the chief boric-acid supply of this country was gathered from the bottoms of desiccated ponds in California and Nevada.

A score of years then passed before it was surmised that the marsh-deposits might be possibly naturally leached from the clay-formations which bordered the dry lakes. The clay-beds themselves then began to be exploited. The utilization of the boraciferous Tertiary clays continued from 1890 to 1905. Some boric acid is still obtained from this source.

In the newest period great deposits of very pure borate-minerals have been found imbedded, or interstratified, in old Ter-

tiary sediments; and large mining-operations have been already started which bid fair to control the borax industry of the world.

The Tertiary clays in which borate-minerals are found occur principally in southern California, but partly in Nevada. They extend in a rather broad belt, semicircular in shape, around the southern extremity of the Sierra Nevada and about 60 miles from that range. From Death valley and the Nevada boundary these clays are exposed at intervals through a distance of more than 300 miles, to the Pacific ocean at Santa Barbara, north of Los Angeles. Along the Furnace canyon on the east side of Death valley, in the Amargosa desert, in the low range of mountains north of Daggett, in the Mojave desert, in the Cajon pass of the San Bernardino mountains, and in the Santa Clara valley, which opens eastward from Santa Barbara, fine exposures give insight into the great areal extent of these deposits. For a decade or more Daggett has been the chief source of the borax-supply in the United States, but during the past year the Furnace Canyon and Santa Clara localities have been so extensively developed that the poorer deposits so long worked at Daggett are rapidly being abandoned. Moreover, better and more accessible deposits than any yet mentioned are ready to be developed.

The areal extent and relationships of the principal boraciferous beds are outlined in Fig. 1. In these localities the borate-layers are best exposed in the mountain-sides, where the stratified clay-beds, from 4,000 to 5,000 ft. thick, containing them have been tilted at a high angle and exposed to the erosive processes. Whether or not all of the borate-layers are in the same geologic horizon is not determined. Nor is it known with certainty that the several localities belong in the same geologic province. Presumably the clay-strata are continuous throughout the entire belt.

III. Geology of the Death Valley Borate-Region.

1. *Surface-Relief.*—The principal borate-deposits of the Death Valley region are interbedded with clay and friable sandstone formations of Tertiary age. These beds are best exposed in the Amargosa range, in Death valley, and in the Panamint mountains. This belt of mountain and valley is 125 miles long and

from 30 to 40 miles broad. To the east is the great Amargosa plain, and to the west the Panamint valley.

The district is a part of the Great Basin region, with its myriads of short, lofty mountain ranges, separated from one another by broad plain-like valleys. In this vast region, stretching from the Wasatch range on the east to the Sierra Nevada on the west, are found some of the most remarkable geographic features on the face of the globe.

The Sierra Nevada, rising from 10,000 to 14,000 ft. above mean-tide, constitutes the most conspicuous relief character of

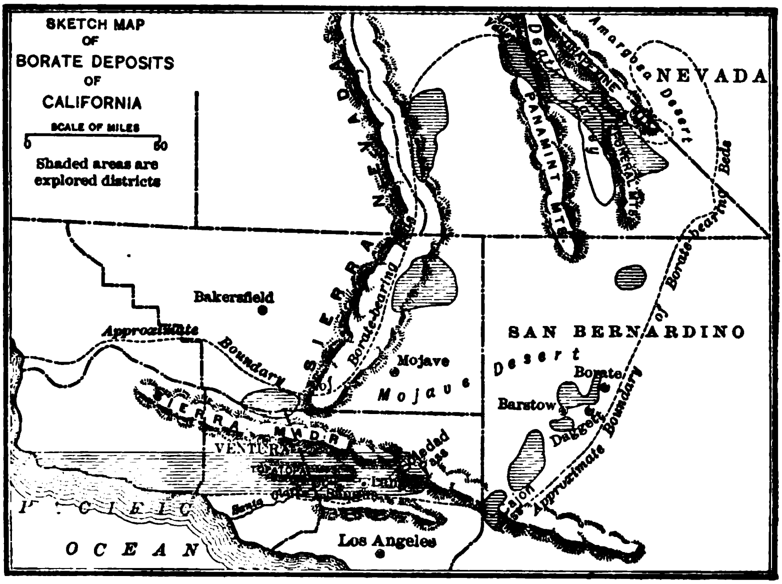

FIG. 1.—SKETCH-MAP OF CALIFORNIA BORATE-LOCALITIES.

the Western United States. Parallel to, and eastward of, this majestic range there are other lesser ridges. The flat-bottomed valleys separating the abrupt mountains from one another are successively lower and lower as the distance increases from the main Sierra until Death valley is reached, which is the lowest depression of all, and the lowest point of any continent. East of the deep valley the intermont plains increase in elevation. The general relief-contrasts thus produced are well shown in profile in Fig. 2.

The immediate area occupied by the soft boraciferous beds

is greatly diversified. The great Amargosa range, in which the principal deposits are found, presents to Death valley a very steep slope, which possibly represents a profound fault-scarp now much degraded. The north and south ends of the range are composed of hard clastics and eruptives separated by a belt of soft, infolded clays and friable sandstones, the belt of the latter passing diagonally across the mountain ridge. This slightly-resistant belt has been worn down from 3,000 to 4,000 ft. below the level of the crest of the range, dividing it into two somewhat distinct and nearly equal parts. The northern portion is usually known as the Grapevine mountains, while the southern part is called the Funeral mountains. The extremities of the Amargosa range fade out into the plains in the same manner

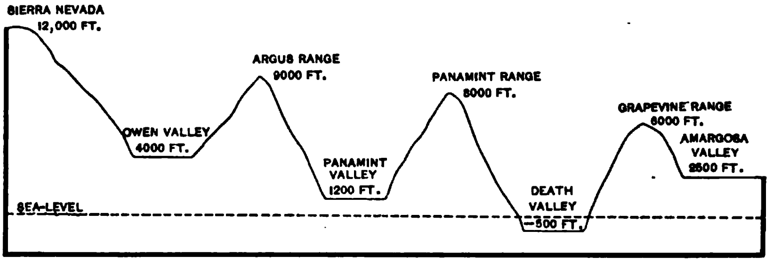

FIG. 2.—PROFILE OF THE DEATH VALLEY BORATE-REGION.

as do the neighboring ranges. It is in the soft middle belt that the chief borate-beds occur. (See Fig. 3.)

The valleys with which the boraciferous beds are particularly associated are the Death valley and the Amargosa valley. Like the majority of the intermont spaces of the Great Basin region and the Mexican plateau, these so-called valleys appear as vast plains seemingly as level as the sea. From the margins on all sides abruptly rise, without intervening foothills,[1] the lofty mountain ranges.

The most remarkable feature of the plains is the general absence of marked drainage-lines. Most of these basin-plains are true *bolsons*, such as are found farther southward in Mexico,[2] while some are *playas*, as the Spanish term them, having broad shallow sheets of water covering their central portions for a part of the year and at other times forming a bare mud-flat.

[1] *Bulletin of the Geological Society of America*, vol. xix., p. 573 (1907).

[2] *American Journal of Science*, Fourth Series, vol. xv., No. 87, pp. 207 to 210 (Mar., 1903).

Many of the intermont plains of the region have beveled rock floors,[3] and they are now believed to be formed chiefly through deflative erosion instead of tectonically, as they were long thought to be.

The drainage-ways leading into Death valley are many, short, and steep, and are occupied by water only briefly at rare intervals of heavy rain-fall in the neighboring mountains, or during the spring melting of the snows. In the mountains these drainage-ways lie in deep narrow canyons, but as they emerge into the great valley they are represented by shallow etchings on broad alluvial fans.

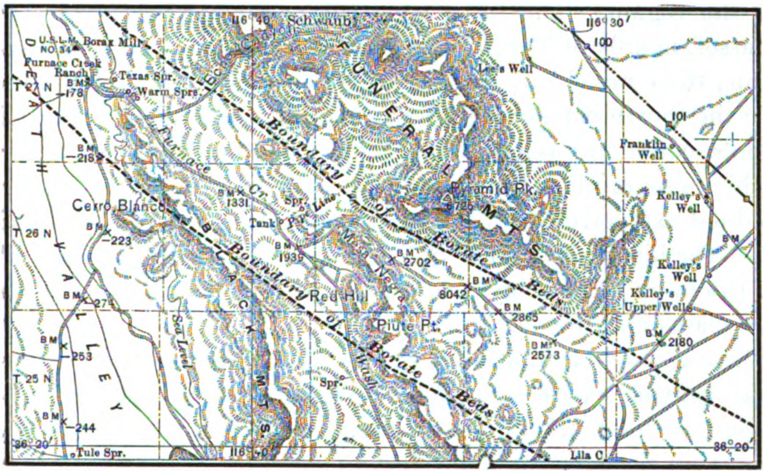

FIG. 3.—MAP OF FURNACE CANYON BORATE-DISTRICT.

Furnace creek has occupied the center of a narrow belt of clays and soft sandstones that traverses diagonally the Amargosa range. This canyon is relatively wide, and of much lower gradient than any of the other drainage-ways of the district. On each side of the main course are numerous deep ramifying canyons, cut in the thick clay-beds, the latter withstanding weathering in a remarkable manner. The crest of the canyon-walls on the east is capped by a thick basalt-flow, which forms a high cliff for a distance of several miles.

2. *Geologic Formations.*—The Death Valley region, which until quite recently has remained one of the least visited portions of

[3] *Bulletin of the Geological Society of America*, vol. xix., p. 67 (1907).

the United States, has never received from travelers more than incidental mention. Aside from a few brief and scattered notes, nothing has yet been published regarding the formations containing the borate-minerals. Altogether the region still remains geologically a veritable *terra incognita*.

The geologic formations of Death valley may be grouped readily into five great classes. The first of these includes certain gneisses and schists, such as appear in the basal part of the Amargosa range north of Furnace canyon, and which are probably of Azoic age. The second group contains hard, and often somewhat metamorphosed, clastics of Palæozoic age; these constitute the foundation of the various mountain ranges of the region. The third class comprises extensive volcanic masses, mainly of Tertiary age, but some of very late extravasation; these are chiefly diorites, rhyolites, andesites, and basalts. To the fourth category belong great deposits of soft clays and sands, commonly regarded as of lacustrine origin, which attain a thickness of more than 4,000 ft., and which often have interbedded extensive basalt sheets. They are largely of Early- and Mid-Tertiary age. With the fifth group may be included all of the more recent clays, sands, and gravels which now mantle the plains and the valleys.

The geologic formations exposed in the vicinity of the Furnace canyon, where the chief borate-deposits are located, represent a total thickness of about 20,000 ft., systematically arranged as shown in Table I.

The fundamental complex, composed of the highly metamorphosed schists and gneisses, is but sparingly displayed in the immediate vicinity of Furnace canyon. The foundation of both the Amargosa and the Panamint ranges comprises mainly quartzites and hard blue limestones of Palæozoic age.

In the neighborhood of the Furnace Canyon pass, at the south end of the Grapevine mountains, the principal part of the mountain ridge appears to be made up principally of Cambrian rocks. A few miles to the north higher beds come in, including peculiar terranes, which seem to be the southward extensions of the Pogonip limestone of King[4] and the Eureka quartzite of Hague.[5]

[4] *Report of the Geological Exploration of the Fortieth Parallel*, vol. i., p. 232 (1878).

[5] *Third Annual Report, U. S. Geological Survey*, p. 254 (1881–82).

TABLE I.—*Geologic Formations About Furnace Canyon.*

	Age.		Thickness.	Rocks.
CENOZOIC.	Quarternary.	Late.	Feet. 500 200	Basaltic flows. Gravels, sands, and clays.
		Early.	1,500	Basaltic flows. Alluvial deposits. Basaltic flows.
	Tertiary.	Late.	1,000	*Playa* deposits.
		Mid.	2,500	Clays and sands with basalt-flows. Clays and marls.
		Early.	4,000	Andesites. Rhyolites. Diorites.
PALÆOZOIC.	Carboniferous.	Mid.	2,500	Limestones.
		Early.	300	Limestones.
	Devonian.	....	100	Limestones.
	Silurian.	....	100 400	Limestones. Quartzites.
	Ordovician.	....	3,000	Limestones.
	Cambrian.	....	2,500	Quartzites.
AZ.	Huronian.	....	500	Gneisses and schists.

Besides the Ordovician rocks, there appear to be represented beds of Silurian, Devonian, and Carboniferous limestones. While the first mentioned are known to be entirely absent from the central portions of the Colorado plateau, and are generally regarded as not being present anywhere around its margins, recent inquiries show conclusively that beds of this age certainly occur at many points in the peripheral belt. In the Amargosa range possibly 400 ft. of limestone seems clearly referable to Silurian age. The exact relations of this section to the Lone Mountain limestone of the Eureka district are as yet undetermined, but it is most likely that the two are not coextensive.

At several places in the Amargosa range fossils have been discovered indicating the presence of Devonian beds. At least 100 ft. of strata is thus tentatively referred to this age. Devonian limestones and shales, long thought to be absent around the entire margin of the great dome of the Colorado plateau, have been found recently to be well represented.

Walcott,[6] for instance, has found rocks of this age in the Grand Canyon district, between the Cambrian Tonto formation and the Carboniferous Red Wall terrane. On the south side of the dome the Devonian beds are highly fossiliferous.[7]

The more recent formations of the region under consideration include three main groups of rocks: the early acid volcanics, the clays and friable sandstones and their associated deposits, and the interbedded basic lavas. The latter are to be clearly distinguished from the basaltic surface-flows of Quaternary age. The Mesozoic formations appear to be entirely absent, unless some of the rhyolites and andesites should finally prove to be partly of pre-Tertiary age.

In the immediate vicinity of Furnace canyon the early volcanic rocks find small exposure. To the northward, beyond Boundary canyon, they make up much of the Grapevine range. These rocks appear as numerous and successive flows of what is commonly termed porphyry. The complete sequence of these acidic lavas is at least 4,000 ft. thick, and consists partly of dull grayish and reddish andesites, but mainly of multicolored rhyolites. The first-mentioned flows are much the older, and may be eventually found to be Jurassic in age rather than Tertiary.

South and west of Furnace creek, in that part of the Amargosa range known as the Funeral mountains, the greater part of the mountains is composed of similar andesites and rhyolites, with considerable dioritic and monzonitic masses. The principal volcanics in the Panamint range on the west side of Death valley also appear to be light reddish monzonitic or granitic rocks.

All of these acidic volcanics seem to have been outpoured over more or less level plains, and the mountains partly elevated before the stratified clays and sands were deposited. The latter, of which the Furnace Canyon borate-bearing deposits may be regarded as typical, are widely distributed. They have a thickness, in this district, of probably more than 4,000 ft., and comprise mainly soft, yellowish to brownish sandstones,

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[6] *American Journal of Science*, Third Series, vol. xxvi., No. 156, p. 438 (Dec., 1883).

[7] *American Journal of Science*, Fourth Series, vol. xxi., No. 124, pp. 296 to 300 (Apr., 1906).

with numerous clay-layers and greenish-yellow to whitish clays. There are a few calcareous beds. The clayey portions of the deposits contain the borate-minerals. Interbedded with the sands and clays are many sheets of basalt, of which the individual layers often are 100 ft. thick.

The most recent terranes consist of, besides the wash from the mountains, some finer deposits of temporary lakes, and perhaps also *playa*-deposits. These are separable into several distinct formations. Some borate-minerals are found in these beds, but thus far the deposits have proved to be unimportant. Extensive basalt-flows of a very late date cover large areas, and in many cases preserve the underlying soft clays from erosion.

3. *Geotectonics.*—Death valley is not only the lowest part of the Great basin, but the lowest area on the whole continent, being 500 ft. below mean sea-level. Contrary to prevailing opinion, the general tectonics of the Basin region is regarded as quite ancient. The so-called Basin Range type of mountain-structure is thought to be the exception rather than the rule, as an explanation of the rearing of the desert ranges. In the recent treatment of the origin of the Basin ranges the present relief-features were viewed from the stand-point of deflation, or wind-erosion, upon a planed surface composed of alternating belts of hard and soft rocks.[8]

The Death Valley district displays perhaps as well as any other area the distinctive characteristics of the so-called Basin Range structure. As it appears on first glance the geologic structure of the Amargosa range is that of a long, narrow, monoclinal block, tilted towards the east; that of the Panamint range, a huge mountain-block inclined westward; that of Death valley, a key-block 10 miles wide dropped down between, forming what the Germans call a *Graben*-block. The general idea is represented in Fig. 4.

Militating against this explanation is the singular fact that no one has yet been able to point out any direct evidences of recent profound faulting along the sides of the *Graben.* This is a significant fact, also noted by Spurr,[9] as applying to the majority of the desert ranges assumed to represent typical so-called Basin Range structures.

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[8] *Bulletin of the Geological Society of America*, vol. xix., pp. 63 to 92 (1907).

[9] *Bulletin of the Geological Society of America*, vol. xii., pp. 217 to 272 (1900).

The marked flexings, faultings, and unconformities displayed in the region, it may be noted in passing, appear to have little direct influence upon the existing relief expression. It is only to the minor, later deformative effects that attention need be specially called in the present connection. While the strata of the region present notable folding, it is so overshadowed by

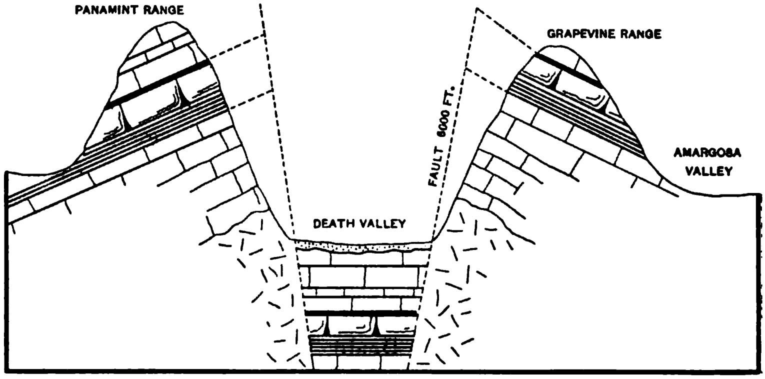

FIG. 4.—GRABEN STRUCTURE OF DEATH VALLEY.

profound and frequent faulting that the phenomenon is not at first glance at all striking.

Some of the more gentle flexures may have been merely attendant or local phenomena of the major faulting. That there is some genetic relationship between the two is further suggested by the fact that the axes of the folds appear to be parallel to the trend of the adjacent mountain ranges. The Tertiary

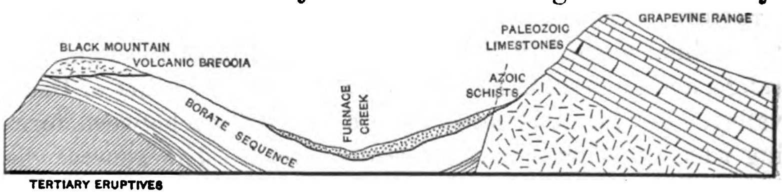

FIG. 5.—CROSS-SECTION OF FURNACE CREEK VALLEY.

clay-strata of the Furnace canyon present the geologic cross-section shown in Fig. 5.

The older major faulting need not be described here, since only the minor faulting has a direct bearing upon the arrangement of the borate-beds. These dislocations have mainly an E–W. trend. They are so recent that in some cases they still impart to the surface-relief a characteristic feature. This is

particularly true when the surface is covered by basaltic flows, as, for instance, on the east side of Furnace canyon, and near the north end of the Funeral mountains.

Discordance in sedimentation is of special importance in considering the deposits of borate-minerals, for the reason that it has a direct bearing upon the economic exploration of the boraciferous field. In the section of more than 4,000 ft. of the Tertiary clays there are a number of great planes of unconformity, besides many minor ones. Some of the latter are so inconsequential as easily to escape notice. At the base of the clays-succession there are abundant evidences of profound unconformable relationships between these deposits and the indurated rocks beneath. Also, in the middle of the sequence, there is a very marked plane of like significance. At the top of the same section is a third great plane of discordance.

IV. Geology of the Borate-Beds of Furnace Canyon.

1. *General Features.*—The principal stratified beds carrying borate-deposits lie, as already stated, in a deep valley and canyon between two lofty mountain ranges. These boraciferous beds consist chiefly of soft clays and sands or friable sandstones, while the mountains on either side are composed largely of hard eruptives and metamorphosed clastics. Under conditions of normal humid climate, and in an elevated region, differential weathering alone would enable the weak formations to be eroded deeply within a very short time. In a dry country the same is also true, except that the erosive agent is chiefly wind-scour instead of water-action.

Furnace canyon, which traverses lengthwise the belt of clays trending diagonally across the Amargosa range, has now nearly bisected the great mountain ridge. This *arroyo*, or "dry-creek," has many lateral branches, deep and labyrinthine. The steep sides are produced partly by the undermining of the thick basalt sheet which still covers and protects from erosion many square miles of the soft clay-deposits. The vertical sections of the beds are many and are finely exposed. For the most part the local attitude of the layered deposits is readily made out.

Three rather distinct phases of the soft stratified beds occur. At the base of the section is a coarse conglomerate, which ap-

pears to be confined in its areal distribution chiefly to the eastern margin of the belt of finer deposits. It may form a part of the lower beds exposed in the Furnace canyon. It is possible, also, that it long antedates the Miocene terranes. For the present, until more critical evidence is obtained, it may be classed with the borate sequence. This conglomerate forms the south end of the Grapevine mountains, on the east side of the Furnace canyon south of Pyramid peak, where it has a thickness of more than 3,000 ft. It is also extensively exposed at the south end of the Funeral range, at the China ranch, where the Amargosa "river," for a distance of a dozen miles, cuts a deep canyon through the formation. At both of the

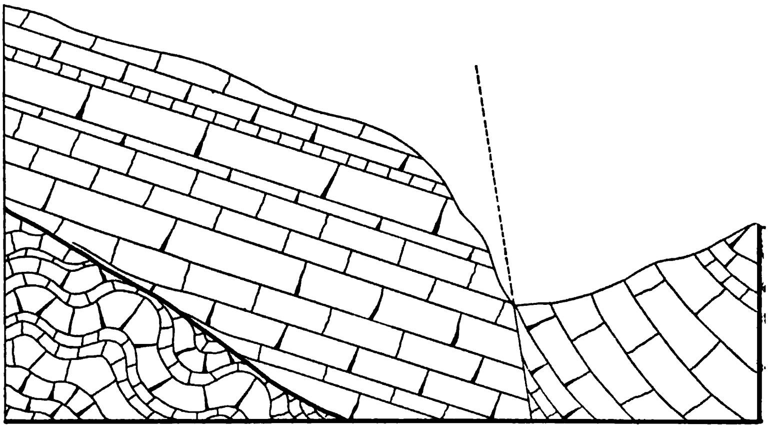

FIG. 6.—SHEARING AT SOUTH END OF FURNACE RANGE.

localities mentioned the terrane is heavily bedded, strongly tilted SE., and nowhere markedly flexed.

The pebbles and boulders comprising the chief portion of the conglomerates are all water-worn and rounded, and are evidently derived mainly from the Palæozoic rocks. The general color is brownish or reddish. There are some fine-grained sandstone-layers. The entire succession of beds is firmly cemented. The exact stratigraphic relationships of the conglomerate with the Palæozoics of the district are not as yet clearly understood. There are marked unconformable relations in some localities, but in one place at least, south of the Pyramid peak, there is a notable shearing, indicated in Fig. 6. Along the thrust-plane the conglomerate-beds contrast sharply with the contorted Ordovician limestones.

The second marked phase of the Tertiary stratified deposits has been called the older sand-series. The section is composed chiefly of yellow to reddish sandstones, rather massively bedded. These sandstones, so far as observed, nowhere merge into the underlying conglomerates; neither are there within them any conglomeratic facies. There are interstratified some minor clay-layers. The upper surface of the sandstones is very uneven and is apparently sub-aërially eroded. The higher clays seem to lie upon them unconformably. The sandstones are well displayed in the Furnace canyon, about 15 miles above its mouth. The thickness of the beds probably exceeds 1,000 ft. Neither in the sandstones nor in the conglomerates beneath are there any indications of the presence of borate-minerals.

The third, and uppermost lithologic, phase consists of fine, alkaline, olive-green clays, which weather to pale yellow or white. Numerous olivine-basalt sheets, from 10 to 100 ft. thick, are interbedded. In the upper part of the sequence is much crystalline gypsum (selenite), thick beds of crystallized calcium borate (colemanite), and thin layers of limestone, probably of chemical origin. Thick beds of rock-salt are also reported to occur in several localities. So far as known, no fossils are found in any of the formations within the area under consideration.

2. *Geologic Structure.*—The strata in the Furnace canyon are all more or less disturbed. The tilting of the beds appears to be chiefly the result of late flexing. The main lines of dislocation, if such they be, blocking out the clay-deposits, strike nearly NW–SE. The main anticlinal flexure trends more nearly E–W. The axis runs nearly parallel to the north branch of the Furnace creek, or Black canyon, and 4 or 5 miles from it. A line drawn from the Lila C. mine to the Cerro Blanco, a distance of 25 miles, nearly coincides with this axis. It crosses the canyon obliquely, making a noteworthy feature of the local topography. The geologic cross-section, SW. from Pyramid peak, is represented in Fig. 7.

From the central core of the older sandstones, as displayed in the Furnace canyon, the younger succession of deposits containing the principal borate-beds dip on either side of the axis in opposite directions. On the east side of the canyon, basaltic lava-flows cover the soft clays, and rest upon their evenly-

beveled edges. The principal borate-beds have been cut through by the excavation of the canyon. Towards the crest of the Funeral mountains they are steeply upturned.

Near the summit of the older axis, where it passes beneath the basalt-sheet at Piute point, the beds are so beveled that the

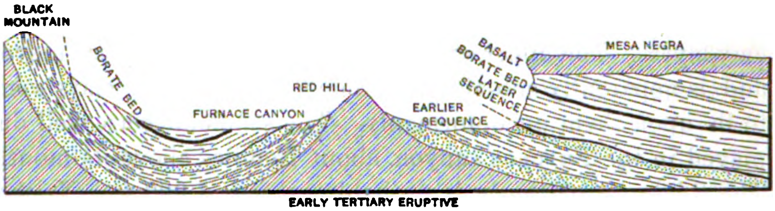

FIG. 7.—CROSS-SECTION OF FURNACE CANYON BORATE-DEPOSITS.

separated ends of the main borate-ledge on either side of the fold are 2 miles apart. The details are shown in Fig. 8.

Two miles to the north of the last-mentioned locality, under the Black mesa, where, near the top of the escarpment, the borate-beds again appear, the latter are sloping to the NE., as shown in Fig. 9.

In the Cerro Blanco, at the north end of the Funeral mountains, the succession of the borate-beds is finely displayed. An E-W. cross-section of the tilted strata is shown in the sub-

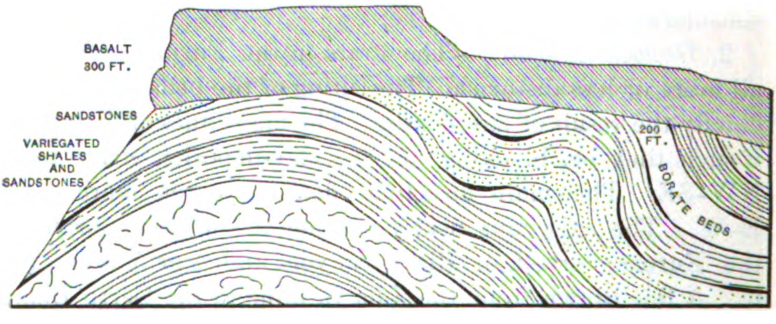

FIG. 8.—ANTICLINAL STRUCTURE AT PIUTE POINT. HEIGHT OF SECTION, 1,000 FEET.

joined diagram, Fig. 10. A notable feature of this sequence is the interbedded basalt-flows. The total thickness represented exceeds 3,000 ft. for the stratified clays alone. There are besides extensive deposits of old gravels, clays, and eruptives in thick sheets. The clay-deposits and sandstones present an alternation of soft and hard layers, which, with their present

attitudes, form a series of sharp parallel ridges separated by deep valleys. In this section also there appear at least two marked planes of unconformity.

3. *Ores.*—The richer borate-beds are from a few inches to

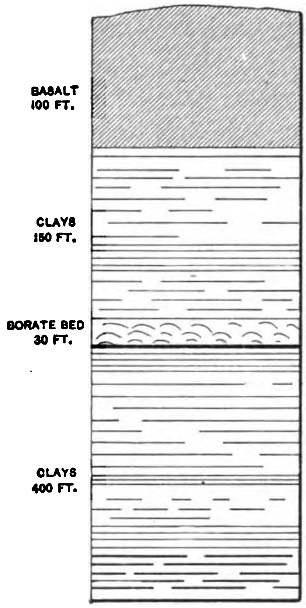

FIG. 9.—FACE OF BORATE-BEDS UNDER THE BLACK MESA.

50 ft. thick. In the unweathered portions they consist of bluish clays thickly interspersed with milk-white layers, nodular bands, and nodules of crystallized colemanite, or calcium borate, which is termed locally "high-grade ore."

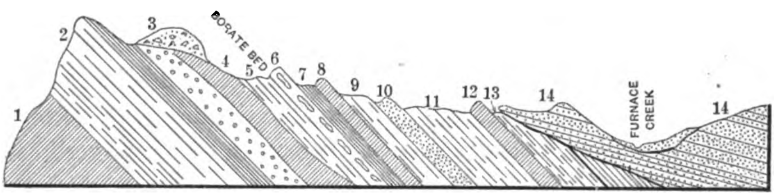

FIG. 10.—TILTED BORATE-BEDS AT CERRO BLANCO, FURNACE VALLEY. SECTION 2 MILES LONG.

Through the strata carrying the coarse, crystallized colemanite, the clays are more or less highly impregnated with fine particles of the borate-mineral, and yield, upon leaching, from 10 to 25 per cent. of anhydrous boric acid. This material is

called by the miners "low-grade ore." While there are large quantities of this low-grade material in the Furnace Canyon district, it cannot be utilized to advantage in competition with the richer layers adjacent to it. At Daggett, however, similar clays carrying as low as from 6 to 12 per cent. of boric-acid content in a finely-divided form are exclusively mined on a large scale by two of the principal borax-producers.

Mingled with the coarse colemanite are often large amounts of crystallized gypsum (selenite) in large plates. In places the gypsum becomes so abundant that the borate-minerals are all but completely obscured. Frequently, also, there are present large amounts of very pure lime, which sometimes forms compact bands resembling layers of ordinary limestone. There are to a greater or less extent associated in the borate-beds other alkaline salts. Special attention will be called to some of these in another place.

There appear to be in the Furnace canyon several distinct horizons at which the colemanite was deposited, but, so far as now known, there is only a single level at which the mineral was formed in large quantities. This horizon is exposed on both sides of the Furnace valley. Large quantities of the mineral have been already removed through the excavation of the canyon.

The clays associated with the borate-beds are all very fine and are entirely free from coarse material, the crystallizations excepted. These clays, when freshly exposed, are blue in color, but on weathering soon become olive-green, then yellowish, and finally nearly white. They seem to be of typical *playa* origin, very much the same kind as are being formed at the present day in many inclosed basins throughout the arid region.

4. *Typical Section.*—In their full thickness the later clay-deposits, or boraciferous beds, are nowhere exposed. The most complete section, showing the changes in the lithologic sequence, is displayed at Cerro Blanco, at the north end of the Funeral mountains. The relationships of the various beds are indicated in Fig. 10. The details of the stratigraphic succession are as follows:

Section of Borate-Beds at Cerro Blanco.

		Feet.
14.	Clays and gravels, pale reddish-brown and purple,	500
	Unconformity, very marked.	
13.	Clay, shaly, argillaceous, yellowish,	200
12.	Basalt, black, surface-flow,	30
11.	Clay, shaly, pale yellow,	500
10.	Sandstone, friable, red in color,	25
9.	Clay, shaly, yellow to green,	150
8.	Basalt, surface-flow,	100
7.	Clay, shaly, olive-green to yellow,	60
6.	Clay, shaly, colemanite in large nodules and nodular layers,	50
5.	Shale, argillaceous and sandy, buff,	300
	Unconformity.	
4.	Basalt,	200
3.	Gravels, coarse, little or no clay,	300
2.	Clay-shale, blue above, yellowish below,	1,000
1.	Andesite (exposed),	500

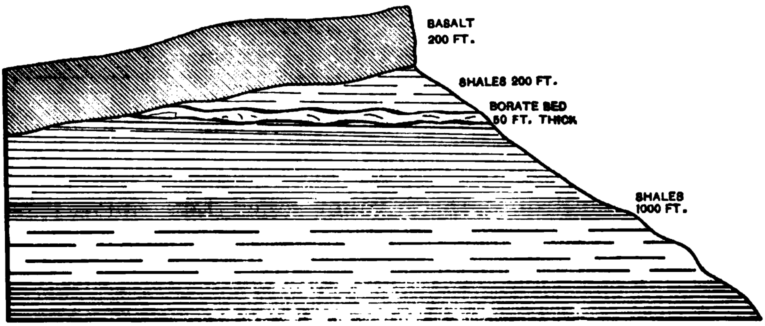

FIG. 11.—BORATE-BEDS UNDER MESA NEGRA, FURNACE CANYON.

On the opposite side of the Furnace valley, and 5 miles SE., under the Mesa Negra, the beds are inclined as shown in Fig. 11. While the thickness of the boraciferous bed is very clearly displayed, the subdivisions of the clay sequence are not so well shown as at the Cerro Blanco.

Two miles south of the last-mentioned locality, under the Black mesa, at a high sharp promontory called Piute point, the section is finely presented, as shown in Fig. 12.

At a mine-opening, 10 miles SE. of the Piute point, on the edge of the Amargosa plain, the clays are inclined about 20° E. In the sides of a low hill, where the borate-bed has been drifted upon, the richer colemanite-bearing stratum is 4 ft. thick, as illustrated in Fig. 13. Although the surface of the ground is obscured by the soil mantle, it is quite evident that

the workable stratum is thicker. The full section is well exposed near the mine-entrance.

V. Borate-Deposits of Lost Valley.

The great belt of yellow Tertiary clays traverses not only the Amargosa range but also the Death valley, and extends

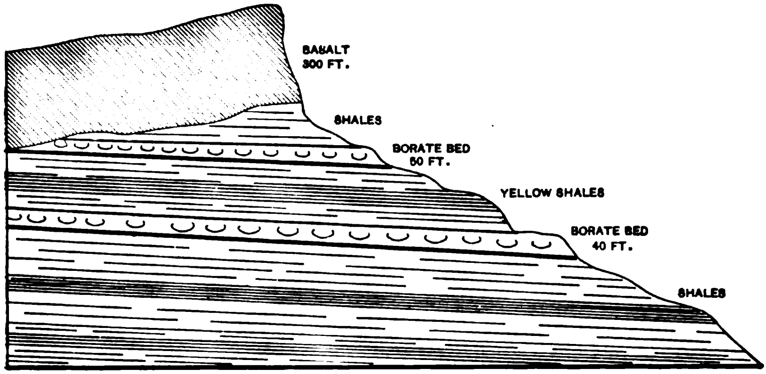

Fig. 12.—Details of Borate-Beds at Piute Point.

into the northern end of the Panamint range. On a spur of the latter, in an arm of Death valley called Lost valley, these clays are finely developed. At a point 25 miles from Cerro Blanco, in a NW. direction, borate-minerals occur. Thus far

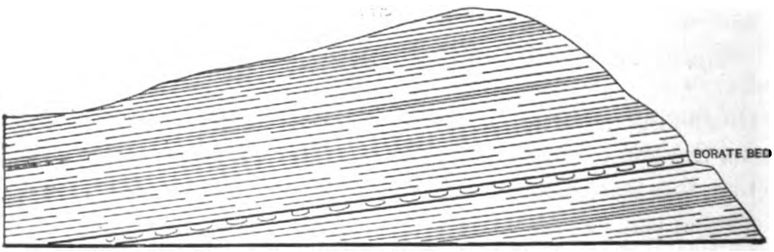

Fig. 13.—Drift on Inclined Borate-Bed at Lila C.

the deposits discovered consist mainly of low-grade material similar to the disseminated beds at Daggett. On account of the inaccessibility of the region at present little systematic exploratory work has been done.

VI. Borate-Deposits in Mojave Desert.

1. *Distribution.*—The peculiar yellow clays and sands, with which the borate-minerals are particularly associated, are widely

distributed in the Mojave Desert region. Only in the vicinity of Daggett, a station on the Atchison, Topeka & Santa Fé railroad, have the borate-minerals been carefully explored and opened up. It is this locality which has furnished, for a period of more than a dozen years, practically all of the borax obtained in this country.

From the crest of the Sierra Madre, at the Cajon and Soledad passes north of San Bernardino, Cal., to the Furnace canyon, in Death valley, the yellow clays are exposed at frequent intervals in the low mountains which protrude above the broad expanse of the Mojave desert. Along the Mojave river, from the Cajon pass north to beyond Daggett, a distance of 75 miles, the outcrops of the formations in question are almost unbroken. They also occur on the opposite side of the desert-basin, on the flanks of the Sierra Nevada, 100 miles north of the Cajon pass. Since on the plains the yellow deposits are often found a few feet beneath the surface-mantle of wind-drifted soils, it is very probable that the same beds underlie the greater portion of the Mojave desert, especially the belt 100 miles wide extending from Daggett to Death valley and beyond.

The borate-bearing deposits are usually spoken of as lake-beds. Upon what grounds I do not know. Lithologically, they appear to be the same from Death valley to the Pacific ocean. Only in the western part of the Mojave plain have fossils been found, and these are marine Eocene and Miocene types. It seems probable that if strictly marine beds extend this far from the Pacific into the Mojave area, the Death Valley beds are also deposits of the sea rather than of extensive lakes in the process of desiccation.

The yellow clays of the Mojave region also contain inter-bedded basalt-flows similar to those occurring in Death valley.

2. *Geologic Structure.*—As admirably shown in the low mountains north of Daggett, the borate-beds are somewhat flexed and frequently infolded with the old volcanic sheets which once were surface lava-flows. The axes of the flexures are mainly E–W., and parallel to the trend of the great Sierra Madre line of uplift on the south.

At the mining-camp of Borate, 12 miles north of Daggett, the inclination of the strata varies from 15° to 50° southward. As shown in Fig. 14, the soft clays have not been deposited

around the foot of the mountains as recent lake-beds, but dip at a high angle directly into them near their summits. The crest of the range is formed by a thick sheet of eruptive rock, which constitutes a protecting cap for the weak clays beneath. The peculiarities of desert eolation permit differential erosion to go

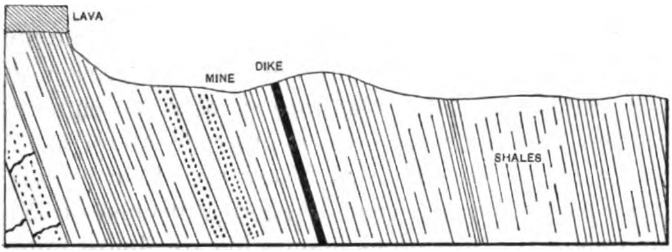

FIG. 14.—VERTICAL TERTIARY BORATE-BEDS NEAR DAGGETT, CAL.

on more rapidly in the arid country than in a normal moist climate.[10] It is estimated that under conditions of aridity erosion of soft rock-masses proceeds ten times as fast in a dry country as it does in a moist one; while under similar climatic circumstances the wasting away of hard rock-masses goes on only one-tenth as rapidly.

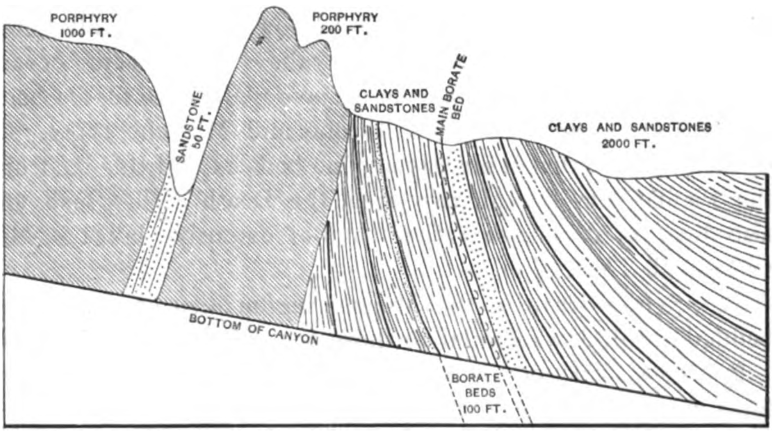

FIG. 15.—BORATE-DEPOSITS AT LANG, CAL.

The surface of the broad valley west of Borate and Calico and 10 miles NE. of Barstow, is over many square miles a true rock floor but thinly veneered by soil. The strata are highly tilted and evenly beveled. Wherever the more-indurated layers

[10] *Bulletin of the Geological Society of America*, vol. xix., pp. 63 to 92 (1907).

occur long ridges are found. At the mines of the American Borax Co. the section is finely displayed, as shown in Fig. 15. The beds dip 75° N. The yellow clays and sands are here more than a mile thick. There appear to be two well-defined borate-horizons separated by about 80 ft. of sandy shale; 15 ft. above the superior bed is a thin andesitic sheet scarcely 2 ft. thick. A short distance south of the mines is a high flat-topped hill. This is composed of the soft clays and sands standing on edge, evenly truncated and covered by a basalt-sheet.

South of Daggett and Barstow, a distance of from 8 to 10 miles, the yellow clays and sands appear in force in several prominent E–W. ridges. These deposits have been prospected for borate-minerals, but have not as yet yielded workable bodies. The strata are only slightly inclined, seldom more than 5° or 10°. The soft yellow formations have interbedded numerous sheets of basaltic and andesitic lavas. In the tilted condition these faulted and resistant layers lying over weak deposits give rise to the long, sharp ridges. To the west these pronounced relief-features gradually melt away into the general surface of the immediate valley of the Mojave river, indicating that the lava-flows do not extend far in that direction.

3. *Ores.*—At the mines of the American Borax Co., NW. of Daggett, there are, as already noted (Fig. 15), in the yellow-clays section two distinct horizons from which the borate-materials are obtained. Both beds are about 5 ft. thick. The mineral is in a finely-divided state, the blue clay of the beds worked containing 10 or 12 per cent. of anhydrous boric acid. The 80 ft. of clays separating the two productive layers contain some borate-material, but not enough to make it profitable at the present time to remove. Near the old Calico gold-mine, 6 miles east, another borax-refinery is obtaining its crude material from similar deposits. There are doubtless other horizons in the general section which are borate-bearing.

At the mining-camp of Borate there are also two workable beds, about 50 ft. apart. Whether or not these two levels are the same as those worked at the American mine is not at the present time known. The high-grade borate here is mainly the calcium salt occurring in nodular crystallized masses scattered through the blue clays. This nodular colemanite is now mined at depths of from 400 to 500 ft., and is treated at the refinery

near Daggett. The associated low-grade material out of which the crystallized colemanite is separated is not at present utilized, although elsewhere it is the low-grade boraciferous clays that are leached.

A photographic view of the borax-mines at Borate is given in Fig. 16; the refinery and the evaporation-racks of the American Borax Co., at Daggett, in Figs. 17 and 18, respectively; and Fig. 19 shows the general type of concentrating-vats used in the Mojave valley.

VII. Borate-Deposits of Santa Clara Valley.

1. *Distribution.*—Yellow sands and clays of Tertiary age are extensively involved in the foldings of the Sierra Madre extending from the Cajon pass to the Pacific ocean. The Santa Clara valley follows the southern foot of the Sierra, and eastward separates it from the western end of the San Gabriel range. On the north side of the valley the nearest range of the Sierra Madre is known as the Topatopa mountains. Near the eastern extremity of the latter, a few miles NE. of the junction of the two branches of the Southern Pacific railroad at Saugus, and 5 or 6 miles NW. of Lang station, important borate-deposits have been recently discovered.

The boraciferous formation is one of great thickness, variously estimated at different places at from 5,000 to 8,000 ft. It comprises mainly fine gravel-beds, more or less indurated, yellow sandstones, and yellow clays. These are traversed by intrusive masses. Judging from the outcrops at the south end of the railroad-tunnel under the Fernando pass, in the San Gabriel range, the beds immediately inclosing the borate-deposits near Lang appear to belong to the Vaqueros terrane, lately described by Eldridge.[11] This is of early Miocene age. Other parts of the Lang section may belong to the Pliocene Fernando formation of the same writer.[12] The geologic structure of the Santa Clara valley is complicated. The strata are profoundly disturbed, so that the detailed relationships of the formations in different parts of the region are not easily grasped without extensive investigations.

Compared with the yellow boraciferous clays of the Mojave desert and Death valley, the Santa Clara section contains much

[11] *Bulletin No.* 309, *U. S. Geological Survey,* p. 12 (1907). [12] *Ibid.,* p. 22.

Fig. 16.—Borax-Mines at Borate, Cal.

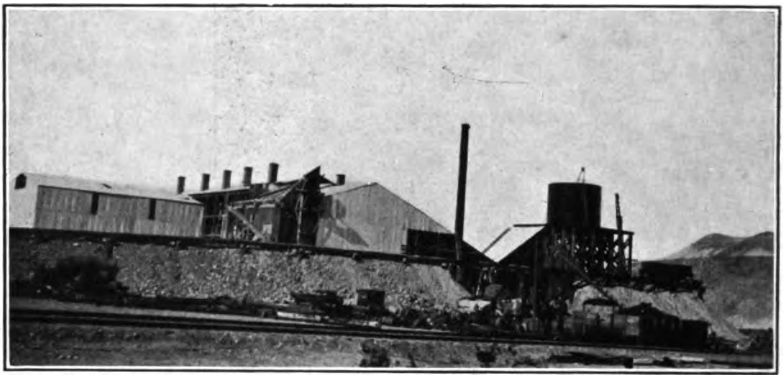

FIG. 17.—REFINERY OF THE AMERICAN BORAX CO., AT DAGGETT, CAL.

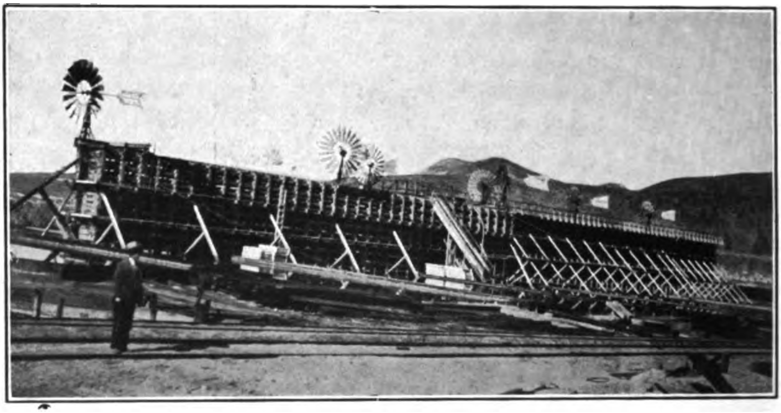

FIG. 18.—EVAPORATING-RACKS AT THE REFINERY OF THE AMERICAN BORAX CO., DAGGETT, CAL.

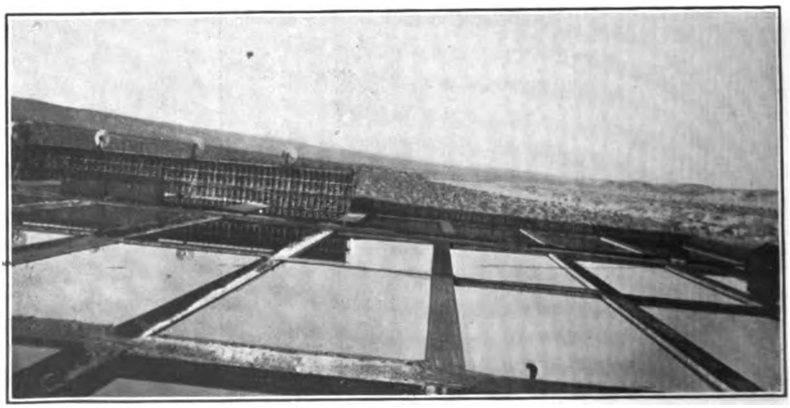

FIG. 19.—CONCENTRATING-VATS IN THE MOJAVE VALLEY.

more sandstone and conglomerate, which suggests that the borate-deposits of the latter district may be of somewhat later geologic age than the former.

2. *Geologic Structure.*—Notwithstanding the recency of formation and the great thickness of the yellow clay and sands in the Santa Clara district, the strata have been severely flexed and profoundly faulted. Several great unconformities tend also to vastly increase the complexity of the stratigraphy. The general tectonics of the Topatopa range, a few miles west of the borate-producing locality, is well displayed in the cross-section near Piru, modified from Eldridge, Fig. 20.

A short distance north of the Lang borate-belt volcanic rocks abruptly take the place of the soft clays and friable sandstones.

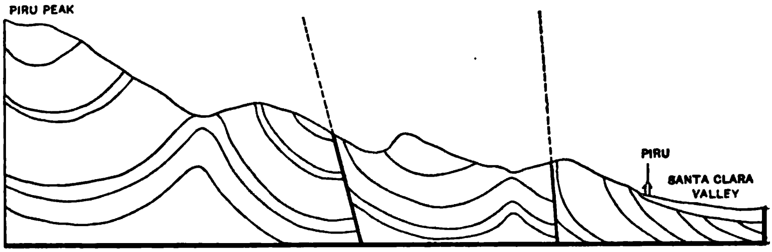

FIG. 20.—GENERAL STRUCTURE OF THE SANTA CLARA VALLEY, CAL.

In these porphyries considerable metal-mining is carried on. The strike of the strata is nearly E–W., but at the borate-mines there is repeated off-setting of the borate-stratum by faulting, having a trend NE–SW., and of several hundreds of feet lateral displacement in each case.

3. *Ores.*—At the point at which the borate-mines are opened the rocks are very much disturbed. The strata are abruptly upturned against a great basic dike, Fig. 15. As observed along the base of the canyon cutting the deposits transversely, the succession of layers is as follows:

Borate-Section at Lang.

	Feet.
6. Sandstone, hard, drab,	25
5. Clays, yellowish, with thin limestone-lenses, and several arenaceous layers,	100
4. Clay, bluish, carrying abundant colemanite nodules and layers,	4
3. Clays and sandstones, yellowish, often calcareous,	200
2. Porphyry dike,	200
1. Sandstone, indurated, brown (exposed),	50

As yet mining is not carried on by means of shafts, as in the cases of the Daggett and Borate localities. In a deep narrow canyon which cuts across the almost vertically-disposed boraciferous bed, and near the bottom, tunnels are driven into the canyon-walls on either side. The mineral is then stoped down, just as the larger ore-bodies are removed in metal-mining. The exceptionally good transportation-facilities, and the high-grade of the material, well combine to make this locality, for many years to come, the principal source of borax in the United States.

VIII. Borate-Deposits in Ventura County, Cal.

On the north side of the Sierra Madre, in the extreme corner of Ventura county, Cal., there occur extensive beds of the yellow clays and sands, similar in all respects to those found on the south slope of the same mountain system and in the Mojave desert. The strata are all highly inclined, and the colemanite is scattered through the blue clay in small nodular crystallizations. The locality has been worked for a number of years, and has furnished a considerable amount of material for the manufacture of boric acid. Since this point is 75 miles from the railroad, the operation of the mines has been conducted under great difficulties, but these in large measure are soon to be removed.

IX. General Geologic Occurrence of Borax.

1. *Original Sources.*—As a title in commerce the name borax is usually applied not only to the substance borax, $Na_2B_4O_7$, itself, but to the several boron compounds which are capable of being readily converted into the sodium tetraborate.

The element boron is widely distributed through the earth's crust. Commonly, however, it is found in such small quantities as to be almost inappreciable. As all of the boron salts, from which the article of commerce is derived, are quite soluble under ordinary climatic conditions, no valuable deposits of these salts occur in normally moist lands. As geological deposits the salts of boron are possible only under climatic conditions of extreme aridity.

Sea-water is known to contain minute amounts of borax. According to Forchhammer,[13] boron is one of the 27 elements the presence of which he detected in the waters of the ocean.

[13] *Philosophical Transactions of the Royal Society*, vol. clv., p. 208 (1865).

Certain of the rock-forming minerals have boron as an essential constituent. Its vapors are regarded as highly important mineralizers in the metamorphism of rocks and in connection with the formation of many ore-deposits.

Boric acid is a common exhalation accompanying volcanic eruptions. Vulcano, Stromboli, Etna, Vesuvius, and other active volcanoes in different parts of the globe give it forth in notable amount.

Many warm and brine springs give forth waters carrying in solution very appreciable quantities. The waters from the great Ash Meadows springs, in SW. Nevada, contain so much borax as to be noticeable to the touch. Roth,[14] especially, has called particular attention to the presence of borates in some of the brine-springs of Germany.

In the drier regions of the globe many of the bitter-lake waters contain considerable amounts of borax. This has been concentrated through long-continued evaporation. In some of these shallow lakes the borax forms in well-defined but scattered crystals in the muds of the bottom. In the old Tertiary clays of the West borates appear to have originated in large deposits in this way.

Geologically, deposits of borax derived from four of the five original sources just enumerated are unimportant. While formerly most of the borax of commerce was obtained from solfataric vapors and from the evaporation of strongly saline waters, little from these sources is now collected. At the present time the greater part of the world's supply of borax comes from the arid regions, where alkaline lakes in the last stages of desiccation yield either borax direct or borates from which it may be artificially derived.

2. *Solfataric Borax.*—Boric acid occurs as a sublimate in lava-cavities and cracks around active volcanoes. Acidic magmas in cooling give off such appreciable amounts of boric vapors that these, together with those of fluorine, chlorine, etc., become important "mineralizers" of the rocks through which they pass. So early as 1846, Élie de Beaumont, the famous French geologist, emphasized the activity which must be displayed by such vapors as those of boron, phosphorus, and fluorine in being

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[14] *Allegemeine und chemische Geologie*, vol. i., p. 442 *et seq.* (1879).

expelled from consolidating granite magmas.[15] Since that time others have expressed similar views.

It is now a well-established fact that the borate-producing localities of the world are also districts in which volcanic activity has not yet ceased. The extent to which boron compounds occur at these places may be judged from the statement that the vapors in some situations are collected in commercial quantities, as in the Maremma of Tuscany. The vapors, as they issue from the *saffioni*, are passed through vats of water, which eventually become charged to the extent of 2 per cent. with boric acid, when the waters are drawn off and the process repeated.

3. *Lacustrine Borax.*—Lake-waters containing small percentages of boric acid have yielded some of the principal borax-supplies. In the United States the most noteworthy of such occurrences are at Clear lake, in northern California, and at Ragtown lake and Sand springs, in Churchill county, Nev.

At the Clear Lake locality the crystals of boric acid occur abundantly in the muds of the bottom of the lake. These muds are pumped out, washed, and sent to the refining-plant. The waters are also boiled in small vats and the boric acid finally crystallized. In Nevada the lake-waters were pumped out upon a plain and allowed to evaporate in the dry air.

4. *Marsh Borax.*—From the *playas* of the arid regions the major part of the borax-supply was formerly obtained. The bottoms of desiccated lakes are often, for a part of the year, covered by a few inches of water. The alkaline crusts which gather upon the floors upon complete evaporation of the waters are harvested and sent to the refinery. The material thus obtained is usually the native borax, $Na_2B_4O_7$, mixed with a number of other salines.

Half a century ago, when the principal portion of the world's supply of borax came from Thibet, in central Asia, it was from such lake-floors that the unrefined material was chiefly gathered. In the United States the main supply was for many years obtained in a similar way. Searle marsh, in the NW. corner of San Bernardino county, Cal., has long been noted for this class of borax.

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[15] *Bulletin de la Société géologique de France*, Second Series, vol. iv., p. 1249 *et seq.* (1846–47).

It is generally assumed that such salinas as Searle marsh are the final remnants of former extensive lakes. According to the latest observations and deductions concerning the evolution of desert relief-features, it seems more probable that the majority of such salinas are due directly to the fact that eolian erosion has encountered ground-water level, permitting their constant evaporation just at the surface of the ground without forming open bodies of water.[16]

5. *Terranal Borax.*—Borates forming old geological deposits are now known to occur in a number of places in the arid regions. The layers, imbedded with shales and sandstones, are associated with gypsum, rock-salt, and other salines deposited from desiccating bodies of water. The bedded borates of California are the most important deposits of the kind known. They form geologic terranes in the strictest sense of the word. It is from this source that the world's supply in the future may be expected mainly to come. The laborious harvesting of lake-muds and thin surface-crusts will soon be a thing of the past. Borax-gathering now becomes a strictly mining industry.

As more fully stated in another place, large bodies of water are known to have existed in very recent geologic times in many parts of what are now eastern California and western Nevada. The smaller of these inland seas, for some of them were cut off from the ocean, soon became bitter-lakes, and finally dried up altogether.

As such bodies of water pass from the stage of saline lakes to that of complete desiccation many interesting precipitations take place. According to the most recent investigations on the subject, ordinary gypsum begins to be deposited on the lake-floor when about 37 per cent. of the water has evaporated. Then as the water progressively reaches the point of saturation for other salts they are thrown down in turn. Finally, when 93 per cent. of the water has passed off, common salt is deposited.

The most frequent succession of salts thrown down by progressive evaporation of saline waters of inland seas is: 1, boracite; 2, anhydrite; 3, gypsum; 4, sylvite; 5, halite, 6, kieserite; 7, polyhalite; 8, kainite; 9, carnallite; 10, tachyhydrite.

Contrary to long-accepted opinion, the various salts in saline

[16] *Bulletin of the Geological Society of America*, vol. xix., p. 90 (1907).

waters under conditions of arid climate are not precipitated in inverse order of their solubilities. The relative amount of the several elements in solution has a prime influence. This differs widely in different basins, so that under the same climatic conditions the same succession of salts does not always appear. Time is a noteworthy determining element. Temperature also plays an important rôle; and pressure has some influence.

The notable factor to be taken into account in considering the general sequence of the salts thrown down in bitter-lakes is the early appearance of the borates.

X. Chemistry of Natural Borates.

1. *General Considerations.*—Since the borates which supply commerce with most of the raw materials for conversion into borax as it is used in the arts now come from old lake-beds of inland-sea deposits, their chemical relations and development are quite like those of saline deposits generally. While a general sequence of salts in the precipitations from complex saline waters has been commonly regarded as established, it is now known that this succession is not everywhere invariably the same. Neither is the sequence in inverse order of solubility, as it was long thought to be.

The experiments on evaporating large quantities of sea-water carried on many years ago by the celebrated Italian scientist, Usiglio,[17] are well known. The results obtained by this chemist have been widely accepted; but more recent tests prove that they are not of so wide application as was at first supposed. Careful comparisons show that the artificial processes do not correspond exactly to the natural ones. This fact recently led the German chemists, Van't Hoff, Meyerhoffer, Hindrichsen, and Weigat,[18] to conduct exhaustive researches on the salt-formations in nature. Very interesting results were obtained, which throw a flood of light upon the subject, and offer satisfactory explanations to many hitherto little understood phenomena.

Among the important factors which Usiglio, and others who have been especially interested in similar experimentation, did not take into consideration were: 1, the composition of the

[17] *Annales de Chimie et de Physique*, Third Series, vol. xxvii., pp. 92 to 107 (1849).
[18] *Sitzungsbericht der königlich preussischen Akademie der Wissenschaften*, 1897.

saline waters; 2, the solubility of the compounds present; 3, the time allowed for concentration; 4, the temperature at which saturation for a given salt took place; and 5, pressure under which crystallization began. Since the recent chemical results have such a direct bearing upon the saline deposits under consideration, they may be briefly summed up here.

In the great salt-deposits of Stassfurt, Germany, which were chiefly investigated, it was found that in the succession of strata four very distinct zones were recognizable. These, beginning at the bottom and named after the principal salt found in them, were the anhydrite zone, the polyhalite zone, the kieserite zone, and the carnallite zone. In all of these zones rock-salt is found. There are also other salts present which are regarded as of secondary formation.

The desiccated inland-sea deposits of the Great Basin region of western America have not been as yet investigated in detail to determine the full variety of salts and their relationships. However, sufficient is known in the case of the borate-deposits of the Death Valley district to state something regarding the peculiar conditions existing at the time at which the salts belonging to the first or lowest zone were precipitated. This zone is the one containing, besides anhydrite, the borates, gypsum, calcite, and some other salts in which lime is an important constituent.

2. *Composition of Saline Waters.*—Were it merely oceanic waters with which we had to deal the chemistry of natural salines would be very simple. By not taking into account the calcium salts the composition would be identical the world over. The composition of the waters of bitter-lakes is very much more complex and varied. Many new conditions are introduced. Inclosed bodies of water, especially those of the very dry regions of the earth, receive compounds in solution from the surrounding elevations that vary greatly in every case, and according to the composition of the rocks, or geologic terranes. In every known instance some one salt greatly predominates.

Instead of the various salts being precipitated in inverse order of solubility, it appears that in a given solution the component which is greatly in excess is the one that is most likely to reach the point of saturation first, and hence will be the first to crystallize out. As Van't Hoff has recently clearly shown,

concentration will continue until the water reaches the point of saturation for a second salt, when that also will commence to be precipitated. If for the moment we can neglect the other salts, in order to give the problem its simplest form, it is from this point onward that the water remains with the composition unchanged. The water gradually evaporates and the salts continue to fall until complete desiccation has taken place.

3. *Solubility of Components.*—There is a widespread opinion among scientists that the salts which crystallize out of saline waters in the arid regions of the globe are merely in solution, and that merely the proper point of concentration is required to be reached in order to precipitate a given salt. Such, it has been already intimated, is not really the case.

Recent observation has conclusively shown that in the desiccation of some saline waters certain salts which naturally would be expected to be found do not appear at all. In other cases compounds entirely unexpected are actually deposited. Under one set of physical conditions the waters of bitter-lakes as they evaporate may throw down a certain series of salts, while under slightly different conditions the same saline waters may deposit an entirely distinct series of compounds.

The first-mentioned results are rather unduly emphasized on account of their being the outcome of laboratory-experimentation also. Here the physical conditions are always very nearly uniform, and the methods of chemical procedure fixed. In nature there is no such uniformity of conditions as is found in the laboratory. In consequence there are many departures from the artificially-conducted tests. Solubility is also a function of temperature, and varies in degree very greatly, as all laboratory-work shows.

4. *Time-Element in Water-Concentrations.*—In nature the time-factor in the determination of precipitates in solution is probably very much more important than is commonly assumed. In the chemical laboratory time is of necessity practically eliminated in all experimentation, and as a consequence very erroneous conclusions are often drawn regarding the chemical processes at work in the earth's crust and the results attained.

The unexpected chemical reactions in nature are as noteworthy in the desiccation of saline waters as they are among the rock-magmas in the process of solidification. Among the

last mentioned granite alone may be cited out of the many known examples. It is shown that an acidic magma, owing to the presence of aqueous vapor, the high pressures under which alone granite can form, and the long time that must manifestly pass, may cool down considerably below the temperature required to crystallize out certain minerals under ordinary dry-fusion conditions. Thus quartz, which should be formed quite early in the normal sequence, can be the last to crystallize, solidifying the whole mass into solid rock. This principle was long ago formulated by Scheerer,[19] who later advocated it at greater length and in greater detail.[20] It was subsequently confirmed experimentally by Èlie de Beaumont, Daubrèe and others, as well as by some more recent investigators.

In the case of similar retardations in crystallization of salts in saline waters under much simpler conditions than those existing among molten materials, recent inquiry has clearly indicated that such phenomena occur very much more frequently than was ever surmised. Length of time, however, is not the only determining factor in these cases.

In the formation of natural salts in desiccating lake-waters the time-factor must be regarded as of prime importance. To it must be ascribed the presence in the sequence of saline deposits of certain salts which never appear in the laboratory-trials. Concerning the saline deposits of the old inland seas of the Great Basin region, this time-factor explains much that previously was very obscure.

5. *Effect of Temperature.*—The general influence of temperature in effecting the crystallizations in saline solutions need not be dwelt upon at length here. Effects of the high temperatures are now well known. Effects of slight changes of a few degrees, within the limits of the ordinary temperatures as they are known in saline waters of the arid regions, have not been so well understood or considered.

At normal temperatures saline waters of the desert basins may deposit a certain number of salts and in a certain sequence. Under conditions of 20° or 30° increase waters of identical composition in the process of desiccation may give rise to some

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[19] *Poggendorff's Annalen der Physik und Chemie*, vol. lvi., pp. 479 to 505 (1842).

[20] *Bulletin de la Société géologique de France*, Second Series, vol. iv., p. 468 *et seq.* (1846-47).

entirely new minerals. At the same time, at the higher temperature, some of the salts which commonly appear at lower degrees of heat do not form at all. Within certain limits the salts derived from evaporation of the waters of saline or bitter-lakes may be regarded as indices of the temperatures of the waters at the time the deposits took place. Hence, it is possible to use deposits of this kind as factors in the determination of geologic climate.

Temperature of saline waters has also a very important bearing upon the paragenesis of many of the minerals which are commonly associated in old lake-beds or deposits of inland seas. The gathering of winter and summer sodas in some of the alkaline ponds of Wyoming and elsewhere forms a good illustration. Just what part temperature has played in the formation of the borate-deposits of California has not yet been definitely determined, but it is thought to be highly influential.

6. *Effect of Pressure.*—The effect of pressure in the formation of saline materials in saline lake-waters can hardly be so great as it is in the cases of many other geologic deposits. Variations in pressure must be quite negligible, because the bodies of water of this kind are comparatively shallow when the salts begin to form. In laboratory-experimentation pressure is usually eliminated altogether.

XI. Occurrence of Other Commercial Salines.

The Tertiary boraciferous formations of southern California are the most remarkable and most extensive in the world. They were formed under conditions of an arid climate in a great shallow arm of the Pacific ocean that had been cut off by the upheaval of the mountain ranges along the coast. The inland sea was long in drying up, and perhaps had frequent connection with the ocean, as is shown by the enormous thickness of the terranes carrying the borates. The disappearance of the water may have been more rapid than the thickness of the deposits suggests at first thought, for the reason that as an accompaniment of the evaporation of the waters in an excessively dry climate there may have been a filling-up of the basin by the prodigious quantities of wind-borne dust derived from the neighboring deserts. It is not to be inferred that, since the

Tertiary clays and sands are between 5,000 and 8,000 ft. thick, the waters in the beginning were at least of the same depth, but rather that the arm of the ocean and afterwards the inland sea was always very shallow, and that as the area was filling up the waters continued to rest on the surface, rising with the rise of the bottom. This postulates a gradual sinking of the foundations of the region, and the truth of this is indicated by the general tectonics of the region.

The entire field of the Tertiary clays in southern California is capable of great results from systematic prospecting and exploration for commercial salines other than calcium borate. The inferences to be drawn from the modern conceptions of the deposition of salines are that with proper inquiry a large series of natural salts may be discovered. The calcium borate-beds are easily passed over unnoticed unless special care be taken to look for them. Other borates are found even more valuable than the colemanite. Extensive rock-salt deposits are already known, as are those of purest gypsum, anhydrite, and calcite. In some of the bitter-lakes immense bodies of soda and magnesia of the kind known mineralogically as blœdite, are among the most wonderful deposits recently found. In one small lakelet scarcely a mile across it is estimated that more than 1,000,000 tons of this mineral is readily available.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Conditions and Costs of Mining at the Braden Copper-Mines, Chile.

BY WILLIAM BRADEN, NEW YORK, N. Y.

(Spokane Meeting, September, 1909.)

THIS paper is presented in the hope that it will be instructive in view of the future large expansion of the mining industry in the west-coast countries of South America.

There is a more or less general impression that the Spanish-American workman is inferior to the American, but after some years of experience and observation I doubt the correctness of this view. Taking into consideration all the elements which make for efficiency of labor, it has been found, particularly in Chile, that under proper organization native labor yields as much, man for man, and more, dollar for dollar, than in the Western United States.

If a manager is willing to accept unreservedly the *costumbres del pais*, without combating intelligently and patiently the ones tending to inefficiency, he should expect no better results than he deserves. He must conscientiously insist that a "square deal" be given and exacted; that liquor be excluded from camp as far as possible; that comfortable quarters and other uplifting elements of life be provided; and that the *costumbres del pais* shall not be over-ridden roughshod. He should also insist upon the gradual and reasonable adjustment of these conditions to the exigencies of the work; and (by no means the least difficult of his tasks) he should select the most competent men to patiently direct and teach the natives in the several departments, and by their own example encourage self-respect and decency.

While the Braden mine is by no means as yet operated on a large scale, according to modern rating, it will nevertheless be interesting to note what has actually been accomplished there. An illustrative description of the mines and mill of the

Braden Copper Co. has already been published.[1] However, in order to lend value to the figures given, the general conditions will be explained.

The copper-deposit is a zone of mineralized, fractured, and brecciated diorite, 150 ft. wide, lying under a hanging-wall of brecciated tuff, which has an approximate dip of 65°. The mine is dry, and little or no drainage of any kind is necessary. The system of mining may be described as overhead mining on broken ore, in a series of transverse stopes, 10 m. wide, with intervening pillars, 7 m. wide, which will be ultimately extracted by the caving system. The ore is moderately hard, and the roof of stopes, 10 m. wide by 50 m. long, stands perfectly well without timbering. The only timbering in the mine is for the framing of the ore-gates for extracting ore from the stopes.

The air-drills, used to the fullest extent, are handled carefully and intelligently by the Chilean miners, and the results have been so favorable that eventually hand-drilling will be almost entirely supplanted.

Rack-a-rock is the principal explosive used; but dynamite and black mining-powder (the latter manufactured in Chile) are also used to a limited extent.

The ore is trammed out of the mine through a main adit in 1-ton cars by means of a three-phase electric haulage-system (220 volts). It is then carried to the mill, 2,630 m. distant and at 550 m. lower elevation, by two Riblet-system aërial tramways.

Water for the hydro-electric power is supplied through a flume, 30 in. wide by 20 in. high, and about 6,000 ft. long, and a pipe-line from 22 to 18 in. in diameter, which leads to three 42-in. Doble water-wheels, two under 820-ft. head, and one under 840-ft. head, direct coupled to a 200-kw., 2,200-volt alternating-current generator. Current is distributed about a mile and a half to one motor running under the same voltage, belted to drive an air-compressor at the mine, and to various other motors (current transformed to 220 volts) for machine-, blacksmith-, and pattern-shops, and (110 volts) electric sample-driers and electric-light system.

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[1] *Engineering and Mining Journal*, vol. lxxxiv., No. 23, pp. 1059–1062 (Dec. 7, 1907).

The following is an extract from the operating-report for November, 1908. Costs are given in United States currency, and weights in dry tons of 2,000 pounds.

The total cost of breaking 16,185 tons, including superintendence and general charges, was 41 cents per ton.

Upon the basis of 7,304 tons extracted and milled, the costs per ton were distributed as follows:

Ore-breaking,	$0.31
General mine-expense,	0.06
Development,	0.12
Underground tramming,	0.02
Aërial tramming,	0.06
Milling: Operation,	0.27
Repairs (labor),	0.01
Repairs (materials),	0.09
General mill-expense,	0.05
Power,	0.01
Sampling and assaying,	0.05
General expense,	0.23
Taxes, insurance, and interest,	0.04
	$1.32

The ore-breaking account includes much cutting-out for ore-pockets, sub-cross-cuts, raises, and other work of a preparatory nature. More than one-half of the labor was performed by contract. In the stopes, contracts were let on the basis of the number of feet of hole drilled. The actual cost of the labor for one month was:

164 man-days (9 hr.), with 2¾-in. "New Ingersoll" air-drills, drilled 5,082 ft. at a cost of 2.025 cents per foot. Average amount drilled per man-day, 31 ft. 1,008 man-days, by single hand-work, drilled 13,855 ft. at a cost of 8.2 cents per foot. Average amount drilled per man per day, 13.8 feet.

Including all labor, both contractors and day-pay men employed for mining, there were 12 tons of ore broken per man-day.

Since the above date, a system of contracting for ore-breaking by measurement of the ore broken has resulted in a considerable reduction of the cost for that item in the foregoing statement.

Development was in the nature of drifts, cross-cuts, and upraises, with an average section of 4.53 sq. m., and, including all labor, supplies, explosives, mucking and tramming, the cost was $3.54 per foot driven, or 78 cents per cubic meter.

The amount of driving was 379 linear feet, with 876.75 man-days, making 13 linear feet per month per man, or 0.433 ft. per day, and 1.96 cubic meters.

To give a concrete idea of what is considered to be good work, a contract was let for driving a main adit of a section of 6 sq. m., at a total cost of $3.15 per running foot, exclusive of drill-sharpening and repairs; 132.25 ft. were driven in 30 days.

Payment of all labor accounts is made three times in each year, on Jan. 1, May 1, and Sept. 1. The workmen are permitted, however, to draw up to 80 per cent. of their balances at any given time—a custom of the country which leads to constancy of work and gives entire satisfaction. Wherever possible, a bonus is paid to encourage good steady work.

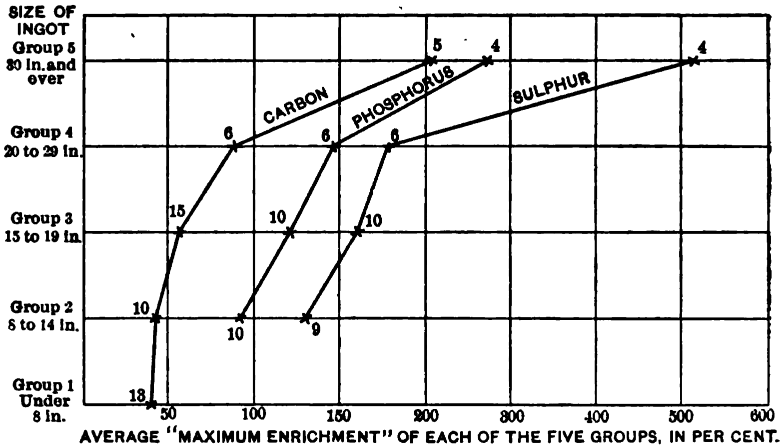
The foregoing data indicate most conclusively the efficiency of Chilean labor.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Influence of Ingot-Size on the Degree of Segregation in Steel Ingots.

BY HENRY M. HOWE, NEW YORK, N. Y.*

THE natural effect of large size should be to increase segregation. I have previously pointed[1] to the excessive segregation in many large ingots as tending to confirm this, but I have shown that in case of ingots less than 16 in. square this expected effect of ingot-size is liable to masked by that of other



NOTE.—The "maximum enrichment" of each ingot, *i.e.*, the excess of the richest spot over the average of the whole ingot, is first calculated in percentage of that average. The average of the maximum enrichment of the several ingots of a given group is the abscissa in Fig. 1. The number beside each spot tells the number of cases which that spot represents.

FIG. 1.—INFLUENCE OF INGOT-SIZE ON MAXIMUM ENRICHMENT IN STEEL INGOTS.

variables. Under these conditions we should expect that if large ingot-size really does tend to increase segregation, this effect would be shown by taking the average of large numbers of cases, so that the effects of these other variables might offset and cancel each other.

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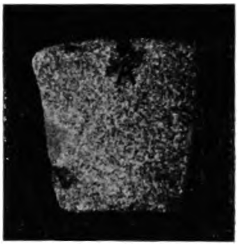
* Professor of Metallurgy in Columbia University, New York, N. Y.

[1] *Engineering and Mining Journal*, vol. lxxxiv., No. 22, p. 1015 (Nov. 30, 1907).

That the degree of enrichment does increase with ingot-size when thus studied is shown by Fig. 1, which represents the average degree of enrichment in 49 different ingots, divided into five classes, according to their size. This figure also brings out prominently the fact that the enrichment in sulphur is greater than that in phosphorus, and that in phosphorus greater than that in carbon. The detailed data on which this figure is based I hope to publish soon. This figure further tends to show that the effect of ingot-size is relatively slight until the thickness of the ingot reaches something like 20 in., but that with further increase of size the enrichment increases more rapidly. This diagram is based on the enrichment at the richest point found in each ingot.

That even very small ingots may be greatly enriched by segregation is shown by Figs. 3, 4, and 5, which represent the microstructure of the neighborhood of a small cavity in the upper part of the axis of a small test-ingot, Fig. 2, only about $\frac{15}{16}$ in. (or 0.94 in.) wide at its widest part and about 5 in. long. It is not necessary to discuss here whether this is a true case of axial segregation or not. My present purpose is to put this interesting case on record. Fig. 5 shows that some of the metal had been enriched in carbon so much as to have turned into white cast-iron, with somewhere about 3 per cent. of carbon. Indeed, the eutectic areas must contain more than 4.3 per cent. of carbon.

The ingot itself is a little acid open-hearth test-ingot, which, after it had sunk to a moderate red heat, was quenched in water as a matter of convenience. Hence the martensitic structure. The metal from the outer part of this ingot contained 1.08 per cent. of carbon by combustion, as determined by J. O. Handy, of the Pittsburgh Testing Laboratory, so that the enrichment even in this minute ingot is not far from fourfold.



A. Cavity.

FIG. 2.—SECTIONAL VIEW OF INGOT. FULL SIZE.

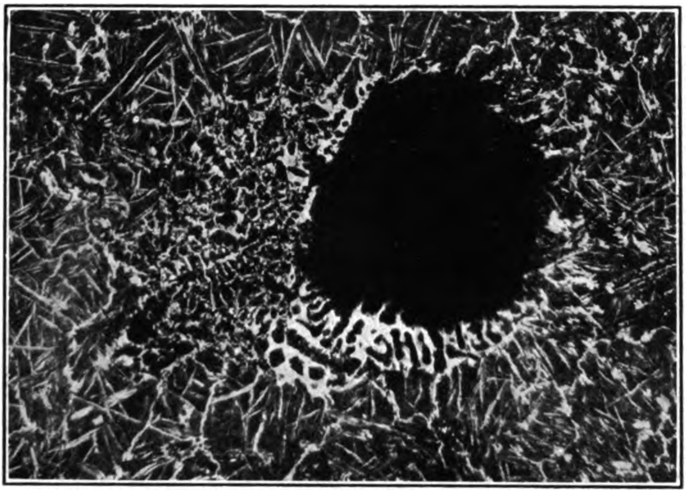


FIG. 3.—CROSS-SECTION THROUGH THE SMALL CENTRAL CAVITY.
MAGNIFICATION, 60/1.

A. Austenite-cementite or white cast-iron eutectic.
B. Primary austenite.

FIG. 5.—CROSS-SECTION THROUGH THE SMALL CENTRAL CAVITY.
MAGNIFICATION, 800/1.

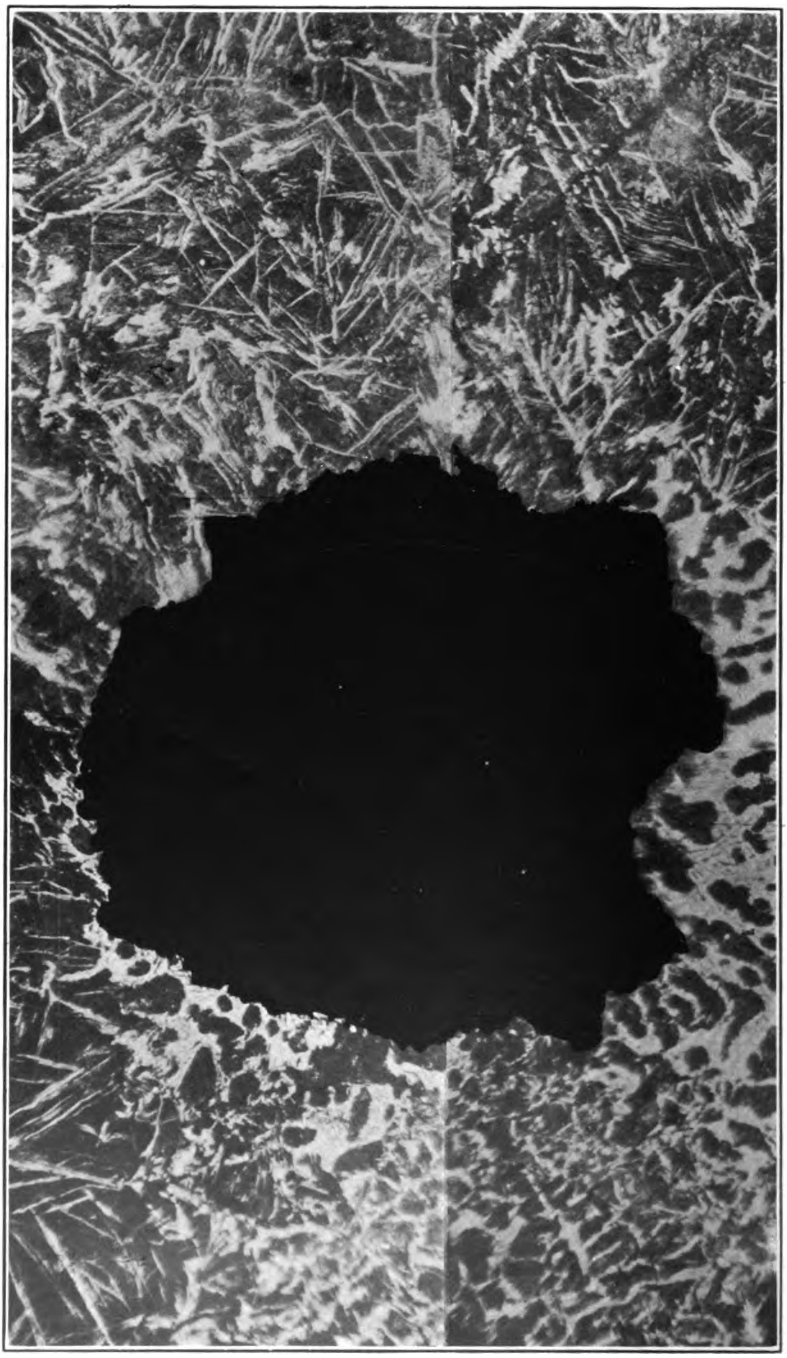


FIG. 4.—CROSS-SECTION THROUGH THE SMALL CENTRAL CAVITY. MAGNIFICATION, ABOUT 130/1.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

The Ruble Hydraulic Elevator.

BY J. McD. PORTER, SPOKANE, WASH.

(Spokane Meeting, September, 1909.)

In many of the old placer-mining districts are still to be found large tracts of gold-bearing gravel not suitable to be worked with a dredge, because the bed is too shallow or the gulch too narrow. Frequently there is not enough grade to handle the gravel successfully by ground-sluicing or a bed-rock flume, or it contains too many boulders to be worked successfully with the ordinary pipe or tube hydraulic elevator.

In southwestern Oregon, two practical placer-miners named Ruble, after working for years trying to make money out of placer-ground containing many large boulders, invented and patented a hydraulic elevator of an entirely new type, and one that has been found to work very successfully in flat ground and in gravel containing many large boulders. It is a very simple contrivance.

A few years ago I acquired the property near Pierce City, Idaho, known as the American placer-mine. Various attempts had been made to work this ground. A bed-rock flume had been installed by one company, an Evans elevator by another, and still other methods were tried on a smaller scale. All were unsuccessful. I installed a Ruble elevator, and it has proved very satisfactory. Working under 100-ft. (pipe) head, the ground has been handled at a cost of a little less than 8 cents per cubic yard. The conditions at the American mine are exceptionally hard, the boulders being large, heavy, and numerous. Basalt-boulders, up to a size of 16 by 18 by 32 in., have been elevated to a height of 16 or 17 ft., with a 4-in. nozzle-stream, under 100-ft. head. Boulders larger than this size are blasted.

At the American mine, a foreman, two pipemen, and two laborers are required per day of 24 hr. to operate the elevator. Several sizes of this elevator are in use, the one at the American mine being the 25-ft. size, having 25 ft. of grizzly

8 ft. wide. This elevator handles from 280 to 300 cu. yd. per day. In ground containing fewer and smaller boulders the capacity would be much greater and the cost of operation much less per day and per yard. At first I used two pipemen on each shift, one driving the gravel to the elevator, the other elevating it, each using a No. 2 giant with 4-in. nozzle. Later, I changed to No. 3 giants, using 5.5-in. nozzles, which enabled me to use all the water in one stream, dispensed with one pipeman on each shift, and materially reduced the cost of operation. I also found that I could handle about 25 per cent. more yardage in this manner than by dividing the water into two streams.

The elevator is so constructed that it is impossible to choke it. The elevator giant, or the giant that elevates the gravel, is placed in line with, and about 50 ft. from, the elevator, which leaves room for the field giant to pile up gravel in front of the elevator. The water is then shut off from the field giant, turned into the elevator giant, and the gravel run through the elevator. While this is being done, the field giant may be moved and reset, if necessary. In deep, undrainable ground, a water-lift is attached to the side of the elevator, to carry off all the water after using. The foreman and laborers can reset the giant while the pipeman keeps the water continually at work. The sluice-boxes may be cleaned up in about 2 hr., while the water is being used in the field giant. The laborers break up the large boulders, cut and burn brush, move and set the giants, help move the elevator, and make themselves generally useful.

The Ruble elevator is a combination of grizzly, undercurrent, and elevator. It separates the fine gravel from the coarse rock and boulders while in transit up the elevator, and does it perfectly. The fine gravel, sand, and gold drop through the grizzly into an apartment underneath the grating-floor, out of reach of the force of the hydraulic stream which is lifting the heavy rocks to the dump. The fine gravel, with its contents, is now acted upon by the water of the elevator giant, after it has lost its force, and with this water is delivered by means of an inclined, smooth floor under the grizzly to the sluice, which is the gold-saving department of the elevator. The fine gravel and sand, after passing the sluice, is delivered to a dump separate

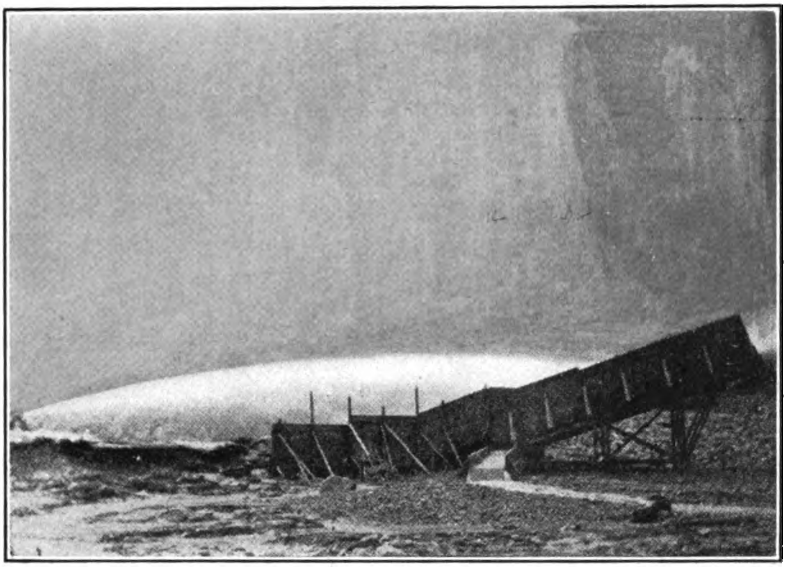


FIG. 1.—THE RUBLE HYDRAULIC ELEVATOR AS OPERATED WITH ONE GIANT.

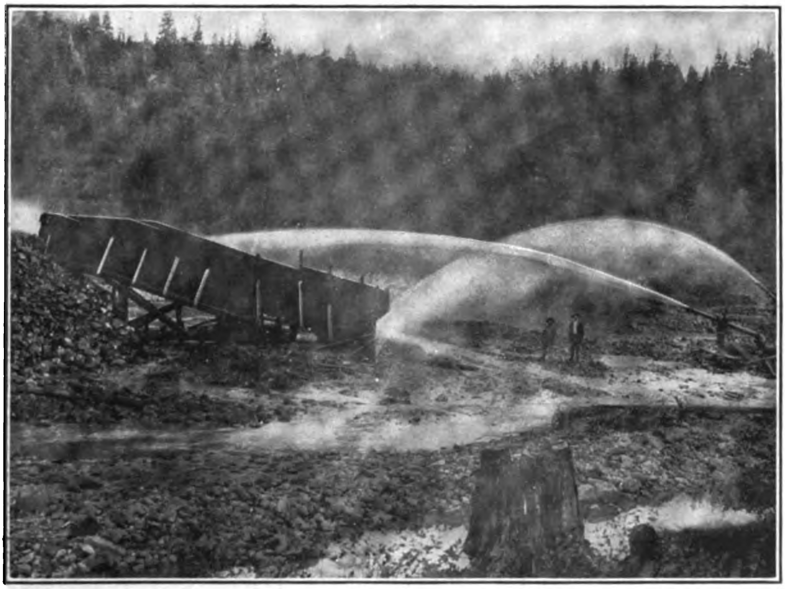


FIG. 2.—THE RUBLE HYDRAULIC ELEVATOR WITH TWO GIANTS IN OPERATION.

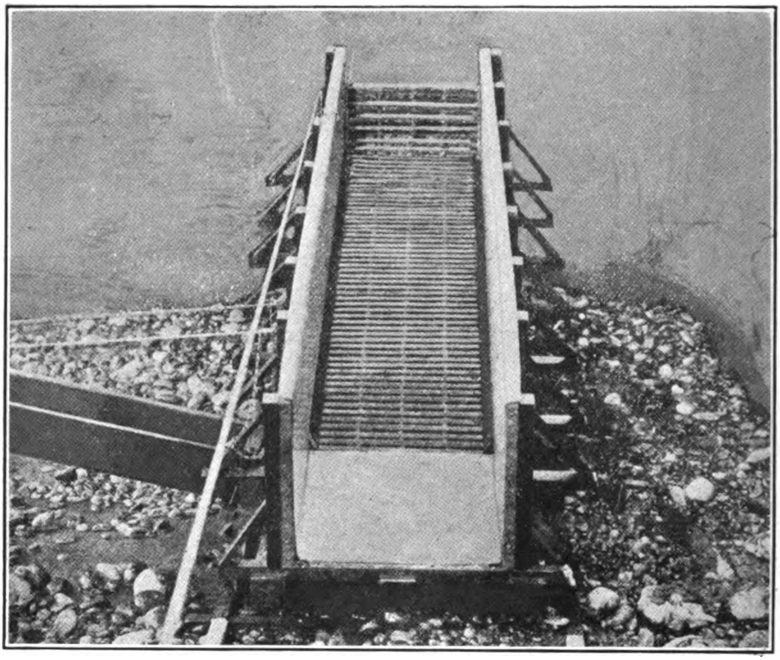


FIG. 3.—GRIZZLY USED WITH THE RUBLE HYDRAULIC ELEVATOR.

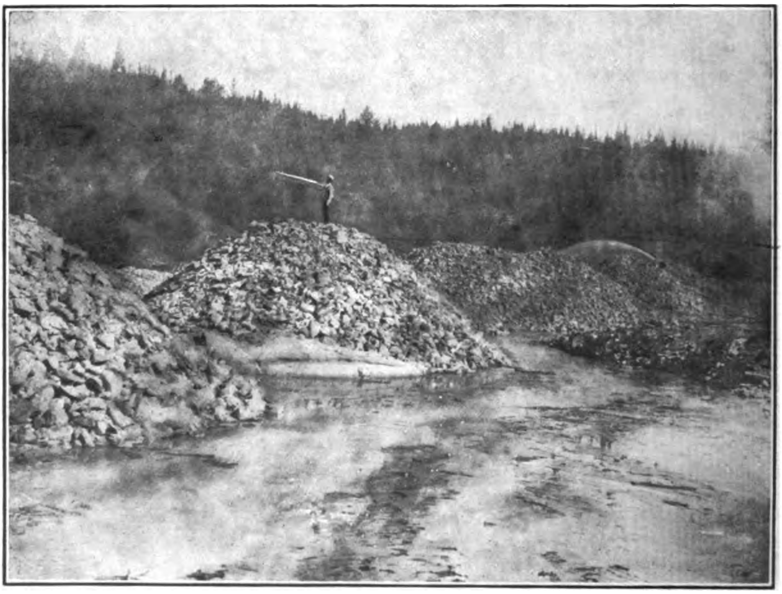


FIG. 4.—BOULDER-DUMPS LEFT BY RUBLE HYDRAULIC ELEVATOR.

from the boulder-dump. Operating with fine material only, the sluices are closely and finely riffled, and are much wider than the usual sluice used for coarse rock, thus furnishing a large gold-saving area in a short sluice.

This appliance solves in an inexpensive and economical manner the difficult problems of inadequate dump, deep and undrainable ground, and the handling of heavy boulders and wash. It effects a close saving of gold and an economical use of water and time.

The elevator rests on rollers, placed on skid-poles on the bed-rock, and is easily moved. When the gravel is washed from in front of the elevator and the space in the rear is filled with boulders and tailings, the bed-rock is cleaned up, a horse is hitched to the elevator, and it is moved forward to a new position nearer the gravel-bank, leaving room for a new boulder-dump in the rear. When horses are not available, the elevator can be moved by hand with a capstan. At the American mine the elevator is moved three or four times each month. It requires from 1 to 1.5 days to move and reset it. The sluice-boxes are usually cleaned up every second day. No time is lost in cleaning up, as the field giant is kept at work during that time.

A few of the points of merit of this elevator are:

1. It handles larger rocks, with less water, than other types of elevators.

2. It dispenses with the boulder-crew, either at the mine, ground-sluice, sluice-box, or dump, except in rare cases.

3. It saves the gold, and the gold-saving department may be protected by lock and key.

4. The gravel is picked up in close proximity to the giants, the gold immediately extracted, and the boulders and other waste material dropped back on bed-rock previously worked off, instead of being transported a long distance to the dump. It makes its own dump.

Strange as it may seem at first thought, it is easier to elevate the gravel than it is to drive it along the level bed-rock, as is shown when the water is divided into two equal streams, the field giant being unable to keep the elevator giant supplied with gravel. In driving the gravel along on the bed-rock, the water has more or less downward pressure, causing more friction, while on the approach and the grizzly, owing to the in-

clined construction, it is lifting the gravel and boulders, causing less friction or dragging.

Fig. 1 shows the operation of the elevator with one giant, the sluice discharging in the foreground.

Fig. 2 shows the elevator with two giants in operation.

Fig. 3 shows the interior appearance of the grizzly; also the sluice swung from its bearings, with the apron or approach removed in order to move ahead.

Fig. 4 shows boulder-dumps left by this elevator at the American mine.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Limit of Fuel-Economy in the Iron Blast-Furnace.

BY N. M. LANGDON, MANCELONA, MICH.

(Spokane Meeting, September, 1909.)

INTRODUCTION.

IN considering the magnificent success of Mr. Gayley's bold experiment of applying dry blast to the blast-furnace, whereby a saving of 20 per cent. of fuel per ton of iron is effected, the question arises whether still further economy in fuel is possible, and, if so, how it is to be attained?

The manner in which the heat generated in the furnace is utilized in the production of pig-iron may be determined, by the method of Bell and Gruner, for any furnace for which the necessary data are obtainable; and the study of the operation, under various conditions, of a number of furnaces, for which such information has been worked out, should enable us to form some conclusion on the subject.

The tables and calculations given below cover the operations of several furnaces reported in our *Transactions;* also the two furnaces using natural and dry blast in Mr. Gayley's experiment, for which the heat-equation has been worked out by Prof. Joseph W. Richards in his admirable discussion [1] of Mr. Gayley's paper, The Application of Dry Air Blast to the Manufacture of Iron, and, finally, various hypothetical furnaces. The full list is as follows:

A. Isabella furnace, with natural blast.

B. Isabella furnace, with dry blast. The data stated for A and B in Table III. are those given by Professor Richards, except that the heat-units are given for one unit of iron instead of 100 units, and some items of heat supplied, as stated by Richards, have been subdivided, the total being the same as given by Richards. (Columns A and B are introduced in Table III. for comparison with C and D.)

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[1] *Trans.*, xxxvi., 745 to 759 (1906).

C. Isabella furnace, with natural blast, for which I have used my own factors in figuring the heat-units.

D. Isabella furnace, with dry blast, for which my own factors have been used for figuring the heat-units.

E. Hypothetical furnace, in which the conditions are the same as in C, except that the temperature of blast is assumed as high enough to reduce the fuel to the same amount as in D.

E1. Hypothetical furnace. Conditions the same as in D, except that temperature of blast is raised to 1,200° F.

E2. Hypothetical furnace. Conditions the same as in C, except that temperature of blast is 1,200° F. Temperature of gas, radiation, and efficiency of reduction the same as in the dry-blast furnace.

F. Hypothetical furnace. Conditions same as in D, except that temperature of blast is assumed at 75° F.

F1. Hypothetical furnace. Conditions same as in C, except that temperature of blast is assumed at 75° F.

G. Hypothetical furnace. Conditions same as in D, except that the moisture of ore is assumed to have been expelled before charging into the furnace.

G1. Hypothetical furnace. Conditions same as in D, except that the CO_2 of limestone is assumed to have been eliminated before charging.

H. Hypothetical furnace, in which the following conditions are assumed: blast dry; no moisture in the ore; no CO_2 in the limestone; temperature of blast, 1,200° F.; temperature of gas, 376° F.; efficiency of reduction, 86.1 per cent. as in C; radiation, 10 per cent. of total heat developed.

H1. Theoretical furnace, with conditions same as in H, except that the moisture of blast is assumed as in C.

I. Theoretical furnace. Conditions same as in H, except that it is assumed that the nitrogen of the blast has been eliminated.

J. Clarence furnace of Bell Bros.[2]

K. Illinois Steel Co.'s furnace, Union No. 1.[3]

L. Sharon furnace No. 2, in which I conducted an experiment covering a period of about two weeks, as I recollect it, in December, 1898.

M1. Alice furnace, Sharpsville, Pa., from data furnished by

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[2] *Trans*, xix., 959 (1890–91). [3] *Trans.*, xx., 287 (1891).

C. I. Rader, manager, in which the efficiency of reduction is assumed at 100 per cent.

M2. Alice furnace, same conditions as M1, except that the efficiency of reduction is assumed at 86.4 per cent., practically the same as in C.

N1. Antrim Iron Co.'s furnace (charcoal), for the month of January, 1901, in which the efficiency of reduction is assumed at 100 per cent.

TABLE I.—*Data Given, Calculated, or (under M1, M2, N1, and N2,) Assumed, for the Furnaces Named.*

		C.	D.	J.	K.	L.	M1.	M2.	N1.	N2.
1.	Capacity, cu. ft.	18,000	18,000	25,500	6,700	....	7,445	7,445	3,416	3,416
2.	Capacity above tuyeres, cu. ft.	16,896	16,896	....	6,424	....	6,666	6,666	3,167	3,167
3.	Hearth area, sq. ft.	148.1	148.1	....	56.7	....	86.6	86.6	33.2	33.2
4.	Output per 24 hours, long tons	358	447	78.6	130	194	236.4	236.4	105.6	105.6
5.	Ore used per ton of iron, long tons.	1.776	1.776	2.40	1.57	1.70	[a]1.662	[a]1.662	2.005	2.005
6.	Limestone, per ton iron, long tons.	0.444	0.444	0.55	0.27	0.589	0.389	0.389	0.151	0.151
7.	Fuel, per ton of iron, long tons	0.958	0.770	1.0	0.779	1.008	0.758	0.758	0.776	0.776
8.	Fuel, per ton of iron, lb.	2,147	1,726	2,240	1,745	2,258	1,698	1,698	1,738	1,738
9.	Temperature of blast, deg. C.	382	465	704	650	560	455	455	427	427
10.	Temperature of gas, deg. C.	281	191	250	125	200	200	200	204	204
11.	Composition of gas by weight.[b] per cent. CO	1.260	0.8925	1.434	0.943	1.2978	[c]0.7922	[c]8801	[c]0.7583	[c]0.8944
12.	Composition of gas by weight.[b] per cent. CO_2	1.155	1.1275	1.095	0.9845	0.9746	1.1725	[c]1.0344	1.192	[c]0.9779
13.	Moisture per cu. ft. blast, grains	5.66	1.75	....	....	....	....	....	....	....
14.	Moisture per ton of iron, tons	0.045	0.01	0.0315	0.0241	0.0368	0.0366	0.0344	0.025	0.023
15.	Composition of pig-iron. per cent. iron	95.0	95.0	....	....	94.4	95.5	95.5	95.5	95.5
16.	Composition of pig-iron. per cent. Si, etc.	1.0	1.0	....	....	2.1	1.0	1.0	1.0	1.0
17.	Composition of pig-iron. per cent. carbon	4.0	4.0	....	....	3.5	3.5	3.5	3.5	3.5
18.	Composition of ore. per cent. iron	53.5	53.5	....	....	55.40	55.4	55.4	47.60	47.60
19.	Composition of ore. per cent. moisture.	10.0	10.0	....	....	8.89	12.0	12.0	10.82	10.82
20.	Composition of ore. per cent. volatile	0.0	0.0	....	....	2.12	3.0	3.0	2.90	2.90
21.	Composition of ore. per cent. gangue	13.6	13.6	....	....	....	....	....	....	....
22.	Composition of limestone. per cent. CO_2	42.6	42.6	....	....	38.84	40.0	40.0	44.0	44.0
23.	Composition of limestone. per cent. moisture	0.0	0.0	....	....	4.60	....	....	4.0	4.0
24.	Composition of limestone. per cent. gangue	57.4	57.4	....	....	....	....	....	....	....
25.	Composition of fuel. per cent. carbon	88.0	88.0	....	....	78.94	86.0	86.0	86.0	86.0
26.	Composition of fuel. per cent. moisture.	1.0	1.0	....	....	5.40	1.5	1.5	6.0	6.0
27.	Composition of fuel. per cent. volatile	0.0	0.0	....	....	1.71	1.0	1.0	6.0	6.0
28.	Composition of fuel. per cent. ash	11.0	11.0	....	....	....	....	....	2.0	2.0

[a] Including 0.102 ton of pig-iron and scrap charged.

[b] In columns A and B (omitted from this table because they are otherwise identical with C and D respectively), the composition of gas is given in percentage by volume as, for A, 22.3 CO, and 13.0 CO_2; for B, 19.9 CO, and 160 CO_2.

[c] Calculated in preparing following tables, and inserted in this table for purposes of comparison.

N2. Antrim Iron Co.'s furnace, with efficiency of reduction assumed at 81 per cent.

Q. Hypothetical furnace. Conditions same as in C, except that the ore is assumed to be protoxide instead of a peroxide.

Q1. Hypothetical furnace. Conditions same as in Q, except that the efficiency of reduction is assumed at 50.8 per cent.

X. Hypothetical furnace, with the following assumptions: charge contains only oxides without slag-making elements;

TABLE II.—*Stock-Equation of Materials Received into and Discharged from Blast-Furnaces. In Tons of 2,240 lb.*

Item.	C. Received.		C. Discharged. Iron.	Slag.	Gas.	D. Received.	D. Discharged. Iron.	Slag.	Gas.
1		Fe..........	0.95	......	0.407 O		0.95	......	0.407 O
2		Si, etc.......	0.01	......	0.011 C		0.01	......	0.011 O
3	Ore......1.776	Gangue.....	......	0.220	....	Ore......1.776	......	0.220	....
4		Moisture...	......	......	0.178 H_2O		......	......	0.178 H_2O
5		Volatile....	......	......	....		......	......	....
6	Limestone,	CO_2..........	......	......	0.190 CO_2	Limestone,	......	......	0.190 CO_2
7	0.444	Gangue.....	......	0.254	....	0.444	......	0.254	....
8		Moisture...	......	......	....		......	......	....
9		Carbon......	0.04	......	0.808 C		0.04	......	0.688 C
10	Coke....0.958	Moisture...	......	......	0.009 H_2O	Coke....0.770	......	......	0.008 H_2O
11		Volatile....	......	......	....		......	......	....
12		Ash..........	......	0.106	....		......	0.084	....
13	Blast...4.392	Air..........	......	......	4.347 air	Blast...3.361	......	......	3.351 air
14		Moisture...	......	......	0.045 H_2O		......	......	0.010 H_2O
15	Total.......7.570		1.00	0.580	5.990	Total........6.851	1.00	0.558	4.793

Item.	E. Received.		E. Discharged. Iron.	Slag.	Gas.	E1. Received.	E1. Discharged. Iron.	Slag.	Gas.
1		Fe..........	0.95	......	0.407 O		0.95	......	0.407 O
2		Si, etc.......	0.01	......	0.011 O		0.01	......	0.011 O
3	Ore......1.776	Gangue.....	......	0.220	....	Ore......1.776	......	0.220	....
4		Moisture...	......	......	0.178 H_2O		......	......	0.178 H_2O
5		Volatile....	......	......	....		......	......	....
6	Limestone,	CO_2..........	......	......	0.190 CO_2	Limestone,	......	......	0.190 CO_2
7	0.444	Gangue.....	......	0.254	....	0.444	......	0.254	....
8		Moisture...	......	......	....		......	......	....
9		Carbon......	0.04	......	0.638 C		0.04	......	0.587 C
10	Coke....0.770	Moisture...	......	......	0.008 H_2O	Coke....0.712	......	......	0.007 H_2O
11		Volatile....	......	......	....		......	......	....
12		Ash..........	......	0.084	....		......	0.078	....
13	Blast...3.440	Air..........	......	......	3.395 air	Blast...3.064	......	......	3.054 air
14		Moisture...	......	......	0.045 H_2O		......	......	0.010 H_2O
15	Total.......6.430		1.00	0.558	4.872	Total......5.996	1.00	0.552	4.444

Item.	F. Received.		F. Discharged. Iron.	Slag.	Gas.	G. Received.	G. Discharged. Iron.	Slag.	Gas.
1		Fe..........	0.95	......	0.407 O		0.95	......	0.407 O
2		Si, etc.......	0.01	......	0.011 O		0.01	......	0.011 O
3	Ore......1.776	Gangue.....	......	0.220	....	Ore......1.598	......	0.220	....
4		Moisture...	......	......	0.178 H_2O		......	......	....
5		Volatile....	......	......	....		......	......	....
6	Limestone,	CO_2..........	......	......	0.190 CO_2	Limestone,	......	......	0.190 CO_2
7	0.444	Gangue.....	......	0.254	....	0.444	......	0.254	....
8		Moisture...	......	......	....		......	......	....
9		Carbon......	0.04	......	0.804 C		0.04	......	0.590 C
10	Coke....0.959	Moisture...	......	......	0.009 H_2O	Coke....0.716	......	......	0.007 H_2O
11		Volatile....	......	......	....		......	......	....
12		Ash..........	......	0.106	....		......	0.079	....
13	Blast...4.319	Air..........	......	......	4.309 air	Blast...3.084	......	......	5.074 air
14		Moisture...	......	......	0.010 H_2O		......	......	0.010 H_2O
15	Total.......7.498		1.00	0.580	5.918	Total........5.842	1.00	0.553	4.269

Item.	H. Received.		H. Discharged. Iron.	Slag.	Gas.	I. Received.	I. Discharged. Iron.	Slag.	Gas.
1		Fe..........	0.95	......	0.407 O		0.95	......	0.407 O
2		Si, etc.......	0.04	......	0.011 O		0.01	......	0.011 O
3	Ore......1.598	Gangue.....	......	0.220	....	Ore......1.598	......	0.220	....
4		Moisture...	......	......	....		......	......	....
5		Volatile....	......	......	....		......	......	....
6	Limestone,	CO_2..........	......	......	....	Limestone,	......	......	....
7	0.254	Gangue.....	......	0.254	....	0.254	......	0.254	....
8		Moisture...	......	......	....		......	......	....
9		Carbon......	0.04	......	0.407 C		0.04	......	0.497 C
10	Coke....0.508	Moisture...	......	......	0.005 H_2O	Coke....0.610	......	......	0.006 H_2O
11		Volatile....	......	......	....		......	......	....
12		Ash..........	......	0.056	....		......	0.060	....
13	Blast...2.073	Air..........	......	......	2.067 air	Blast...0.602	......	......	0.600 air
14		Moisture...	......	......	0.006 H_2O		......	......	0.002 H_2O
15	Total.......4.433		1.00	0.530	2.903	Total.......3.064	1.00	0.534	1.523

Table II.—*Stock-Equation of Materials Received into and Discharged from Blast-Furnaces. In Tons of 2,240 lb.* (*Continued.*)

J.

Item.	Received.		Discharged. Iron.	Slag.	Gas.
1	Ore........2.40	Fe	0.95	......	0.400 O
2		Si, etc.	0.04	......	0.045 O
3		Gangue	......	0.985	....
4		Moisture	......	......	....
5		Volatile	......	......	....
6	Limestone, 0.55	CO_2	......	......	0.242 CO_2
7		Gangue	......	0.308	....
8		Moisture	......	......	....
9	Coke.....1.00	Carbon	0.03	......	0.847 C
10		Moisture	......	......	0.025 H_2O
11		Volatile	......	......	....
12		Ash	......	0.098	....
13	Blast.....4.86	Air	......	......	4.330 air
14		Moisture	......	......	0.082 H_2O
15	Total........8.31		1.00	1.391	5.921

K.

Item.	Received.	Discharged. Iron.	Slag.	Gas.
1	Ore......1.570	0.94	......	0.403 O
2		0.03	......	0.040 O
3		......	0.040	....
4		......	......	[a]0.117 H_2O
5		......	......	....
6	Limestone, 0.270	......	......	0.119 CO_2
7		......	0.151	....
8		......	......	....
9	Coke....0.779	0.03	......	0.640 C
10		......	......	....
11		......	......	....
12		......	0.109	....
13	Blast...3.132	......	......	3.108 air
14		......	......	0.024 H_2O
15	Total.......5.751	1.00	0.300	4.451

L.

Item.	Received.		Discharged. Iron.	Slag.	Gas.
1	Ore......1.700	Fe	0.944	......	0.390 O
2		Si, etc.	0.021	......	0.019 O
3		Gangue	......	0.140	....
4		Moisture	......	......	0.150 H_2O
5		Volatile	......	......	0.036 vol.
6	Limestone, 0.589	CO_2	......	......	0.223 CO
7		Gangue	......	0.339	....
8		Moisture	......	......	0.027 H_2O
9	Coke....1.008	Carbon	0.035	......	0.761 C
10		Moisture	......	......	0.054 H_2O
11		Volatile	......	......	0.017 vol.
12		Ash	......	0.141	....
13	Blast....3.842	Air	......	......	3.805 air
14		Moisture	......	......	0.087 H_2O
15	Total.......7.139		1.000	0.620	5.519

M1.

Item.	Received.	Discharged. Iron.	Slag.	Gas.
1	Ore......1.560 Iron....0.102	0.955	......	0.370 O
2		0.010	......	0.011 O
3		......	0.083	....
4		......	......	0.187 H_2O
5		......	......	0.046 vol.
6	Limestone, 0.389	......	......	0.155 CO_2
7		......	0.234	....
8		......	......	....
9	Coke...0.758	0.035	......	0.617 C
10		......	......	0.011 H_2O
11		......	......	0.007 vol.
12		......	0.088	....
13	Blast...3.552	......	......	3.515 air
14		......	......	[a]0.037 H_2O
15	Total.......6.361	1.000	0.405	4.956

M2.

Item.	Received.		Discharged. Iron.	Slag.	Gas.
1	Ore......1.560 Iron....0.102	Fe	0.955	......	0.370 O
2		Si, etc.	0.010	......	0.011 O
3		Gangue	......	0.083	....
4		Moisture	......	......	0.187 H_2O
5		Volatile	......	......	0.046 vol.
6	Limestone, 0.389	CO_2	......	......	0.155 CO_2
7		Gangue	......	0.234	....
8		Moisture	......	......	....
9	Coke....0.758	Carbon	0.035	......	0.617 C
10		Moisture	......	......	0.011 H_2O
11		Volatile	......	......	0.007 vol.
12		Ash	......	0.088	....
13	Blast...3.346	Air	......	......	3.311 air
14		Moisture	......	......	0.035 H_2O
15	Total.......6.155		1.000	0.405	4.75

N1.

Item.	Received.	Discharged. Iron.	Slag.	Gas.
1	Ore......2.005	0.995	......	0.409 O
2		0.010	......	0.011 O
3		......	0.346	....
4		......	......	0.217 H_2O
5		......	......	0.058 vol.
6	Limestone, 0.151	......	......	0.066 CO_2
7		......	0.079	....
8		......	......	0.006 H_2O
9	Charcoal, 0.776	0.035	......	0.632 C
10		......	......	0.047 H_2O
11		......	......	0.047 vol.
12		......	0.015	....
13	Blast...3.648	......	......	3.623 air
14		......	......	0.025 H_2O
15	Total........6.580	1.000	0.440	5.140

N2.

Item.	Received.		Discharged. Iron.	Slag.	Gas.
1	Ore.......2.005	Fe	0.995	......	0.409 O
2		Si, etc.	0.010	......	0.010 O
3		Gangue	......	0.346	....
4		Moisture	......	......	0.217 H_2O
5		Volatile	......	......	0.058 vol.
6	Limestone, 0.151	CO_2	......	......	0.66 CO_2
7		Gangue	......	0.079	....
8		Moisture	......	......	0.006 H_2O
9	Charcoal, 0.776	Carbon	0.035	......	0.632 C
10		Moisture	......	......	0.047 H_2O
11		Volatile	......	......	0.047 vol.
12		Ash	......	0.015	....
13	Blast...3.313	Air	......	......	3.286
14		Moisture	......	......	0.027
15	Total.......6.245		1.000	0.440	4.805

[a] Assumed.

TABLE III.—*Heat-Equation, or Heat Required and Supplied.*

Item.		A.	B.	C.	D.	E.	E1.	F.	G.	H.	I.	J.	K.	L.	M1.	M2.	N1.	N2.	X.	Q.	Q1.	G1.	J1.	H1.	E2	E3.	F1.	X1.
	HEAT REQUIRED:																											
1	Reduction of iron.........	1,638	1,638	1,791	1,791	1,791	1,791	1,791	1,791	1,791	1,791	1,760	1,773	1,716	1,628	1,628	1,800	1,800	1,791	1,192	1,192	1,791	1,760	1,791	1,791	1,791	1,791	1,791
2	Reduction of silicon, etc.	68	68	67	67	67	67	67	67	67	67	189	211	94	67	67	53	53	67	67	67	67	189	67	67	67	67	67
3	Expulsion of moisture...	113	113	113	113	113	113	113	4	3	4	15	71	141	120	120	164	164	......	113	114	112	10	3	113	112	115	110
4	Expulsion of carbonic acid.........................	180	181	161	161	161	161	161	161	......	......	205	100	189	132	132	56	56	......	161	161	......	101	......	161	161	161	161
5	Expulsion of volatile in fuel....................	......	......	......	......	......	......	......	......	......	......	......	......	38	14	14	94	94	......	......	......	......	......	......	......	......	......	......
6	Expulsion of volatile in ore.......................	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......
7	Expulsion of combined water in ore...	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......
8	Fusion of iron................	250	250	310	310	310	310	310	310	310	310	330	330	330	337	337	300	300	310	310	310	310	330	310	310	310	310	310
9	Fusion of slag................	232	227	290	279	279	276	290	276	265	271	770	165	341	223	223	220	220	......	286	294	275	165	266	279	273	302	259
10	Decomposition of moisture in blast.....	145	32	145	32	145	32	32	32	20	6	102	78	118	118	111	80	72	......	145	145	32	62	74	145	145	145	58
11	*Total direct*...................	2,626	2,509	2,877	2,753	2,866	2,750	2,764	2,641	2,456	2,449	3,371	2,728	2,967	2,639	2,632	2,767	2,759	2,168	2,274	2,283	2,587	2,617	2,511	2,866	2,859	2,891	2,756
12	Carried off in gas..........	438	238	399	217	229	201	268	202	131	69	350	132	260	235	226	247	231	185	365	405	186	224	138	216	188	490	128
13	Carried off in flue dust.[a]	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......
14	Lost by radiation..........	771	630	582	411	422	402	413	387	288	279	405	525	313	591	374	638	301	419	470	476	378	278	295	420	416	602	320
15	*Total indirect*..............	1,209	868	981	628	651	603	681	589	419	348	755	657	573	826	600	885	532	604	835	881	564	502	433	636	604	1,092	448
16	*Total heat required*.......	3,835	3,377	3,858	3,381	3,517	3,353	3,445	3,230	2,875	2,797	4,126	3,385	3,540	3,465	3,232	3,652	3,291	2,772	3,109	3,164	3,151	3,119	2,944	3,502	3,463	3,983	3,204
	HEAT SUPPLIED:																											
17	Carried in by the blast...	394	399	379	370	464	470	24	339	393	93	727	478	509	383	360	368	334	183	368	409	324	442	413	509	693	31	740
18	From C burned to CO at the tuyeres...........	1,846	1,429	1,881	1,454	1,473	1,326	1,864	1,335	902	1,124	1,928	1,420	1,665	1,526	1,434	1,563	1,419	879	1,734	1,928	1,271	1,206	951	1,436	1,213	2,372	754
19	*Total in zone of fusion*...	2,242	1,808	2,278	1,824	1,937	1,796	1,888	1,674	1,295	1,217	2,655	1,898	2,174	1,909	1,794	1,931	1,753	1,062	2,102	2,337	1,595	1,648	1,364	1,945	1,906	2,403	1,491
20	From C burned to CO....	103	121	105	124	105	124	124	124	105	105	167	163	217	0	93	0	144	......	105	248	123	167	105	124	124	105	......
21	From CO burned to CO_2	1,490	1,448	1,475	1,433	1,475	1,433	1,433	1,433	1,475	1,475	1,304	1,324	1,149	1,556	1,345	1,721	1,394	1,710	902	579	1,433	1,304	1,475	1,433	1,433	1,475	1,710
22	*Total in zone of reduction*	1,593	1,569	1,580	1,557	1,580	1,557	1,557	1,557	1,580	1,580	1,471	1,487	1,366	1,556	1,438	1,721	1,538	1,710	1,007	827	1,556	1,471	1,580	1,557	1,557	1,580	1,710
23	*Total heat supplied*.......	3,835	3,377	3,858	3,881	3,517	3,358	3,445	3,230	2,875	2,797	4,126	3,885	3,540	3,465	3,232	3,652	3,291	2,772	3,109	3,164	3,151	3,119	2,944	3,502	3,463	3,983	3,204

[a] This item is relatively small and has been neglected in these calculations.

fuel, pure carbon; temperatures of gas and blast same as in C; iron of the ore is peroxide; blast free from moisture; efficiency of reduction, 100 per cent.

X1. Theoretical furnace, demonstrating the highest theoretical blast-temperature required, with conditions same as in C, so far as they apply, except that efficiency of reduction is assumed at 100 per cent., and temperature of blast as in D.

In presenting data of this kind, it is unfortunate that all writers do not use the same formulas and factors. While the results may be practically the same, comparison is somewhat complicated, and often requires recalculation of the data. In my tables, the data of the Gayley (Isabella) furnaces using natural and dry blast are given first in relative quantities as stated by Richards, and then, together with the similar data worked out for the other furnaces, according to the formulas and factors given by Gruner.

Table I. contains the data given or assumed of the actual furnaces on which the succeeding tables are based.

Table II. shows the stock-equation, or materials received into, and discharged from, each of the furnaces named. All of the materials, in the form of ore, limestone, fuel, and blast, which enter into a furnace come out again in the form of pig-iron, slag, flue-dust, and gas, and, when fully accounted for, make an even balance.

Table III. shows the heat-equation for each furnace. All of the heat generated in the furnace by the combustion of the fuel, or taken in with the blast, is fully accounted for and evenly balanced by the heat consumed in the transformation of the materials, or lost by conduction and radiation.

Table IV. contains data calculated for each furnace named.

Constants.

The following are the constants adopted or calculated from Gruner:

Reduction of iron, . .	weight of O in oxide of iron × 4,400
Reduction of silicon, . . .	weight of O in silicon × 6,125
Fusion of pig, ...	. weight of pig × 310
Fusion of slag, ...	. weight of slag × 500
Expulsion of moisture, . .	. weight of moisture × 606.5
Expulsion of carbonic acid, . .	weight of CO_2 × 849
Decomposition of H_2O, . . .	weight of H_2O × 3,225

Carried off in gas, . .	weight of gas × temp. (C.°) × 0.237
Carried in by the blast, .	weight of blast × temp. (C.°) × 0.237
Heat generated by combustion:	
C to CO, ...	weight of C × 2,473
CO to CO_2, ...	weight of C in CO × 5,607

NOTE.—The weight of O in Fe_2O_3 multiplied by 4,400 is the same as the weight of Fe multiplied by 1,887, as given by Gruner; also the weight of O in silica multiplied by 6,125 is the same as the weight of Si multiplied by 7,000; and the weight of CO_2 in limestone multiplied by 849 is the same as weight $CaCO_3$ multiplied by 373.5.

REMARKS ON TABLES I. TO IV.

The blast-furnace may be considered as a heat-engine, the heat from the fuel and blast being consumed in the transformation of the ore, flux, fuel, and blast introduced into the furnace into pig-iron, slag, and gas discharged therefrom, and the inevitable losses inseparable from the operation.

The total heat consumed or required may be subdivided into a number of different items, and the requirement for each item may be traced to its cause, which may be greater or less in one furnace than in another—or in some cases the cause, and consequently the resulting heat-requirement, may be absent; and as the aggregate heat-requirements made up of the different items may be greater or smaller according to the conditions presented, so may the heat-supply from the blast and fuel likewise vary.

The different items which make up the total heat-requirements are given in Table III., items 1 to 16, inclusive. The total of items 1 to 10 as given in item 11 is the direct heat-requirement. That is to say, the heat for each item is in direct proportion to the quantity of the substance for which it is required, contained in the materials introduced into the furnace, as given in Table II.

The total of items 12, 13, and 14, as given in item 15, Table III., is the indirect heat-requirement. It is affected directly by any variation of the total direct heat-requirement, temperature of the blast, gas, etc.; and is inversely proportional to the efficiency with which the fuel is consumed.

Table III. shows that items 1 to 8, inclusive, of heat-requirement are precisely the same for furnace C, using natural blast, and furnace D, using dry blast.

TABLE IV.—*Data of Operation, Efficiency, Etc., Calculated for the Furnaces Named.*

	C.	D.	E.	E1.	F.	G.	H.	I.	J.	K.	L.	M1	M2.	N1.	N2.	X.	Q.	Q1.	G1.	J1.	H1.	E2.	E3.	F1.	X1.
1. Carbon of CO burned to CO_2 in reducing-zone, tons	0.263	0.2555	0.263	0.2555	0.2555	0.2555	0.263	0.263	0.2325	0.236	0.2049	0.2775	0.2398	0.307	0.2487	0.305	0.1009	0.1033	0.2555	0.2325	0.2630	0.2555	0.2555	0.2630	0.305
2. Carbon of CO possible to have burned to CO_2 in reducing-zone, tons	0.3054	0.3054	0.3054	0.3054	0.3054	0.3054	0.3054	0.3054	0.3000	0.302	0.2925	0.2775	0.2775	0.307	0.307	0.305	0.2033	0.2033	0.3053	0.3000	0.3053	0.3053	0.3053	0.3053	0.305
3. Efficiency of reduction, per cent	86.1	83.7	86.1	83.7	83.7	83.7	86.1	86.1	77.5	78.1	70.0	100.0	86.4	100.0	81.0	100	79.1	50.8	83.7	77.5	87.1	83.7	83.7	86.1	100
4. Carbon burned to CO in reducing-zone, tons	0.0424	0.0499	0.0424	0.0499	0.0499	0.0499	0.0424	0.0424	0.0675	0.066	0.0876	0	0.0377	0	0.0583	0	0.0424	0.1000	0.0499	0.0675	0.0424	0.0499	0.0499	0.0424	0
5. Carbonic oxide (CO) in gas, tons	1.26	0.8925	0.875	0.7726	1.246	0.7805	0.336	0.546	1.434	0.943	1.2978	0.7922	0.8801	0.7583	0.8944	0.1181	1.359	1.811	0.7196	0.7525	0.3830	0.8752	0.6648	1.7239	0
6. Carbonic acid (CO_2) in gas, tons	1.155	1.1275	1.155	1.1275	1.1275	1.1275	0.9643	0.9643	1.0945	0.9845	0.9746	1.1725	1.0344	1.192	0.9779	1.1183	0.7806	0.5694	0.9368	0.9720	0.9643	1.1275	1.1275	1.1550	0.1309
7. Ratio $\frac{CO_2}{CO}$ by weight	0.916	1.263	1.32	1.4594	0.905	1.444	2.870	1.766	0.763	1.044	0.751	1.480	1.1753	1.572	1.0933	9.4691	0.5735	0.3144	1.3018	1.3180	2.5180	1.2800	1.6960	0.6700	0
8. Efficiency of utilization of fuel, per cent	93.5	91.5	92.8	92.3	92.4	91.2	91.3	92.0	91.6	90.1	87.8	100.0	93.9	100.0	90.9	100	92.	83.1	91.0	87.6	91.4	92.6	92.5	94.4	100
9. Active capacity (above tuyeres), cu. ft.	16,896	16,896	16,896	16,896	16,896	16,896	16,896	16,896	....	6,424	....	6,666	6,666	3,167	3,167	16,896	16,896	16,896	16,896	....	16,896	16,896	16,896	16,896	16,896
10. Hearth-area, sq. ft.	143.1	143.1	143.1	143.1	143.1	143.1	143.1	143.1	....	56.7	63.6	86.6	86.6	33.2	33.2	143.1	143.1	143.1	143.1	....	143.1	143.1	143.1	143.1	143.1
11. Ratio of hearth-area to active capacity. 1 to	118	118	118	118	118	118	118	118	....	113	....	77	77	95.4	95.4	118	118	118	118	....	118	118	118	118	118
12. Output of iron per 24 hr., tons	358	447	447	484	359	481	678	564	78.6	130	194	236.4	236.4	105.6	105.6	768	385	328	502	....	647	450	520	290	875
13. Fuel consumed per ton of iron, lb.	2,147	1,726	1,726	1,595	2,148	1,604	1,138	1,367	2.240	1,745	2,258	1,698	1,698	1,738	1,738	[a]886	1,994	2,341	1,537	1,494	1,189	1,707	1,478	2,652	878
14. Rate of driving: lb. C burned in 24 hr. per cu. ft.	36.0	35	35	34.4	36	34	33	34	5.48	26	....	49.0	46.0	47.2	42.9	36	36	34	34	....	33	35	34	37	35
15. Rate of driving: lb. C burned in 24 hr. per sq. ft. hearth-area	4,265	4,118	4,170	4,066	4,240	4,060	3,872	4,016	....	2,948	4,601	3,773	3,542	4,500	4,087	4,277	4,226	4,005	4,041	....	3,900	4,093	4,000	4,358	4,181

[a] Carbon, equivalent to 1,006 lb. coke.

Tabulating the differences shown in Table III., in items 9, 10, 12, and 14 of heat-requirements, and 17, 18, 19, and 20 of heat-supply, between furnaces C and D, we have the figures given in Table V.

TABLE V.—*Comparison of Furnaces C and D (Items* 9, 10, 12, 14, 17, 18, 20, *and* 21, *Table III.), Showing Difference in Heat-Units and Its Percentage of the Total Heat* (3,858 *B.t.u.*) *of C.* (See Item 16, Table III.)

Item.		C. Heat-Units.	D. Heat-Units.	Difference.	Per Cent.
	HEAT REQUIRED:				
9.	For fusion of slag	290	279	11	0.28
10.	For decomposition of moisture in blast	145	32	113	2.93
	Direct saving in heat-requirements by D	....	....	124	3.21
12.	Carried off in gas	399	217	182	4.72
14.	Carried off by radiation	582	411	171	4.43
	Indirect saving in heat-requirements by D	....	....	353	9.15
	Total saving in heat-requirements by D	....	....	477	12.36
	HEAT SUPPLIED:				
17.	Carried in by blast	397	370	27	0.70
18.	From combustion of C to CO at tuyeres	1,881	1,454	427	11.06
	Difference in zone of fusion	....	....	454	11.76
20.	From combustion of C to CO in reducing-zone	105	124	−19[a]	−0.49[a]
21.	From combustion of CO to CO_2 in reducing-zone	1,475	1,433	42	1.09
	Difference in zone of reduction	....	....	23	0.60
	Total difference in heat-units supplied	....	....	477	12.36

[a] In this one item, the figure for D is larger than for C, and therefore is entered with a minus sign.

It will be noted that the furnaces after K in the foregoing list, and in some of the tables, are not discussed in the text. This is due to the circumstance that the tables were originally prepared to accompany a wider consideration of blast-furnace theory, in which the effect of hypothetical changes in dimensions, etc., was to be considered. This more general discussion has been postponed to a future opportunity, so as to base the

present paper chiefly upon data of actual practice. But, at the suggestion of the Secretary, the apparently superfluous columns have been retained in the tables as valuable for further study and reference.

Remarks on Table V.

The small decrease of 11 units in heat required by D under item 9 is due to the saving in total heat-requirements, which involves less fuel, and consequently the presence of less slag-making material, in the furnace using dry blast, as may be seen by comparing item 11 in columns C and D, Table III.

The saving of 113 units under item 10 is due directly to the decreased amount of moisture in the blast of furnace D, using dry air. It will be noted that this saving by itself is only 2.93 per cent., and added to the small item of 0.28 per cent. saved in the fusion of the slag, makes a total direct saving in heat-requirements of only 124 heat-units, or 3.21 per cent.

The saving in heat carried off in the gas (item 12) is 182 heat-units, or 4.72 per cent. of the total heat supplied. This saving is due indirectly to the decreased moisture in the dry-blast furnace D, and directly to the decreased quantity of blast required for the combustion of the decreased quantity of fuel, and the consequent decreased weight of gas discharged; also to the lower temperature of the escaping gas from the dry-blast furnace, which is a direct result of the decreased weight of gas.

In item 14 (radiation), 171 heat-units, or 4.43 per cent. of the total heat-requirements of the furnace using natural blast, are saved by D. It is reasonable to assume that the amount of heat radiated from the furnace is proportional to the temperature and the quantity of heat developed, and that it would vary at different parts of the furnace.

We have in Table III. (item 23) the total heat supplied to furnaces C and D, as well as the portion developed in the fusion-zone (item 19) and the reduction-zone (item 22). The temperature of the fusion-zone, as measured by the composition of the iron and slag, was the same in both furnaces. In the reducing-zone the temperature, as measured by that of the gas discharged, was considerably lower in D (dry blast); therefore this furnace should radiate proportionally less heat than C (natural blast). This is exhibited in Table III., which shows the total loss by radiation in C to be 582 units, or 15.1 per

cent. of the total heat (3,858 units) supplied, while the loss in D was only 411 units, or 12.2 per cent. of the total heat (3,381 units) supplied. In the fusion-zone the loss in C was 582 units, or 25.5 per cent. of the total (2,278 units) supplied to that zone, while in D it was 411 units, or 22.5 per cent. of the total heat (1,824 units) supplied to that zone. Finally, in the reduction-zone, C lost 582 units, or 36.8 per cent. of the 1,580 supplied to that zone, while D lost 411 units, or 26.4 per cent. of the 1,557 supplied. The comparison thus shows a saving by D of 2.9 per cent. of the total heat supplied, 3 per cent. of the heat of the fusion-zone, and 10.4 per cent. of the heat of the reduction-zone.

The difference of 3 per cent. between C and D in the percentage lost by radiation of the heat supplied to the fusion-zone is somewhat above the truth, for the reason that a part of the radiation from D (dry blast) was affected by the decreased temperature of the reducing-zone. It is, however, reasonable to assume that the actual error, if any, is well within the limits of permissible error in calculations of this nature.

The difference of 10.4 per cent. relative to the heat developed in the reducing-zone shows plainly the effect of the cooler top, as well as that of the decreased quantity of heat developed in this zone of the dry-blast furnace.

It is interesting to note in Table V. that the total indirect saving of heat-requirements in the dry-blast furnace is 353 heat-units, or 9.15 per cent.—nearly three times as great as the direct saving; and that, while the total saving in heat-requirements is only 477 heat-units, or 12.36 per cent., yet (according to Table IV.) the actual saving in fuel is 421 lb. per ton, or 19.61 per cent. of the 2,147 lb. per ton of pig-iron used in C with natural blast.

Right here is the *crux* of the question, how and why the direct saving of 2.93 per cent. in heat-requirements for the decomposition of a small quantity of moisture in the blast should effect such a startling saving in the fuel required per ton of iron produced.

Of the fact that there is a saving of about 20 per cent. in fuel there can be no doubt; and many theories have been advanced to account for it.

THEORIES OF THE ECONOMY OF THE DRY BLAST.

Three theories have been advanced, either of which apparently accounts for the saving: "Additional uniformity obtained by the use of dry air," first advanced by Gayley[4] and later emphasized by Raymond[5] and others; "critical temperature," advanced by Johnson[6] and supported by Howe;[7] "more oxygen per day and increased reducing power and smelting-down capacity," advanced by Richards.[8] To these I will add another and very simple reason—namely, that there is less work to be done in the furnace using dry air, and that, as a natural consequence, less fuel is required to do it.

Have those who advance the "uniformity" theory mistaken the shadow for the substance? Applied to the operation of the blast-furnace, uniformity is descriptive of a condition and is not itself a condition. Applied to the moisture of the blast, it is descriptive of its quantity. Taken in this connection it also means uniformly low quantity of moisture. In accounting for the saving of fuel with dry, as compared to natural blast, by the more uniform condition of the moisture of the blast, it is apparent that the full meaning of the term has been lost sight of. There is not the least doubt that uniformity in the moisture of the blast contributes largely to uniformity in the working of the furnace and the condition of its product; but since, in the case of both the natural- and the dry-blast furnaces the working as well as the product is considered at an average, "uniformity" has no significance in accounting for the saving in fuel. Can any one familiar with the phenomena of blast-furnace operation doubt but that, with equal uniformity of the moisture in the blast at its maximum instead of its minimum quantity, there would have been equal uniformity in the working and product of the furnace using moist blast accompanied with higher instead of lower fuel?

Concerning the "critical temperature" theory, there may or may not be such a condition as applied to the working of the blast-furnace. It is evident, however, that there must be a sufficient temperature. From the data of the working and

[4] *Trans.*, xxxv., 771 (1905).
[5] *Trans.*, xxxv., 1023 (1905).
[6] *Trans.*, xxxvi., 472 (1906).
[7] *Trans.*, xxxvii., 216 (1907).
[8] *Trans.*, xxxvi., 745 *et seq.* (1906); xxxvii., 224 (1907).

product of the two furnaces reported by Gayley, we know that the temperature, critical or otherwise, was sufficient in each case. Although we do not know the degree of temperature, we do know, what is of the utmost importance, that it was precisely the same and is accurately measured by the composition of the iron and the slag from each furnace—therefore, this theory will not account for the results obtained by the use of dry blast.

As to more oxygen per day and faster driving, Richards says:[9]

"Weight of oxygen present as air (natural blast), 96.3 kg. per 100 kg. of iron," equivalent to 0.963 ton per ton of iron, which, multiplied by the daily output of 358 tons, gives oxygen per day, 344.75 tons.

"Weight of oxygen present as air (dry blast), 76.5 kg. per 100 kg. of iron," equivalent to 0.765 ton per ton of iron, which, for daily output of 447 tons, gives oxygen per day, 341.95 tons; difference, 2.80 tons.

Or only 0.71 per cent. Evidently this small difference cannot account for the difference in results.

"Increased reducing power" is given as a reason. This is a mistake also. In Table IV. I have given the following conclusions, computed from data given by Richards:

Carbon of CO, Burned to CO_2 in Reducing-Zone, Per Ton of Iron.

	Ton.
Natural blast (Furnace C),	0.263
Dry blast (Furnace D),	0.2555
Difference in favor of natural blast,	0.0075

"Increased smelting-capacity" is the next reason advanced. Right here is the nut; and I think Richards has mistaken the husk for the kernel. The converse of his statement is the true reason; that is to say, less smelting-capacity is required.

In Table V. it is seen how the direct saving in heat-requirement for the decomposition of the moisture of the blast of 113 heat-units, or 2.93 per cent., is increased a little by the slag-smelting requirement, and I have explained how this is still further increased nearly three-fold by the saving in heat carried off by the gases and by radiation, until we have a total saving of 477 heat-units, or 12.36 per cent. It remains to be shown how

[9] *Trans.*, xxxvi., 749 (1906).

this saving of 12.36 per cent. of heat-requirements can effect a saving of 19.6 per cent. in fuel. Item 17, Table V., shows a saving of heat supplied at the tuyeres of 27 units. This is the heat carried in by the blast and does not affect the fuel directly. Item 18, a saving of 427 units, is from the combustion of the fuel to CO and is equivalent in coke to: $\frac{427}{2473} \times \frac{2240}{0.88} = 440$ lb. per ton of iron. In the reducing-zone there were 42 units less developed in the dry-blast furnace from the combustion of CO to CO_2, and 19 units of this must be made up by the combustion of solid fuel to CO, which is equivalent in coke to $\frac{19}{2473} \times \frac{2240}{0.88} = 19$ lb. per ton of iron. The difference, which is saving in coke effected by the use of the dry blast, is 440 — 19 = 421 lb. per ton of iron.

It will be noted that the saving in direct heat-requirements by the use of dry blast is wholly in the smelting-zone, and that, if reduction had been as effective in the dry- as in the natural-blast furnace, the saving would have been 440 lb. of coke.

Notwithstanding the increased temperature of the dry blast, there was, owing to the decreased quantity of blast required per ton of iron, an actual decrease of 27 heat-units carried into the dry-blast furnace, which had to be supplied with coke burned to CO equal to $\frac{27}{2473} \times \frac{2240}{0.88} = 28$ lb. per ton of iron, so that, if all the conditions except that of moisture had been precisely the same in the dry-blast as in the natural-blast furnace, a total saving of 468 lb. of coke per ton of iron might have been effected.

Several writers who have found it difficult to account for the saving of such a large amount of coke as compared with the quantity theoretically required for the dissociation of the moisture eliminated, have evidently assumed, in their theoretical calculations, the complete combustion of the coke to CO_2, overlooking the fact that, owing to the nature of the blast-furnace process, the moisture of the blast would be dissociated at the expense of coke burned to CO, or incomplete combustion, requiring more than 3.25 times as much coke as they deem to be required.

Comparative Results.

The evolution of the blast-furnace, from the primitive form and practice, which consumed more than 7 tons of fuel per ton of iron, to the present form and practice, in which the fuel has been reduced to less than 1,800 lb. per ton of iron, has been marked by five distinct epochs, each of which has produced startling immediate results, as well as permanent changes. The coking of bituminous coal before charging into the furnace, Neilson's application of the hot blast, and Bell's discovery of the relation of the form of the furnace to the proper reduction of the ores, marked three of these epochs. The next profound change, somewhat less startling in its approach and effect, had its maximum development in the wide distribution of the exceptionally rich ores of the Lake Superior region, followed almost immediately by the largely increased supply of artificially-concentrated magnetites. The last epoch of progress, but not the least important in revolutionary results, has been inaugurated by Gayley's process of refrigerating the blast for the purpose of eliminating its moisture.

It is interesting to note that, of the causes above named as contributing to the reduction of the amount of fuel required to make a ton of iron, only one—that of Bell—involves any change in our view of the nature of the blast-furnace process, or of the reactions taking place within the furnace; all the others pertaining simply to a better preparation of the materials (ore, blast, fuel, etc.) before they enter the furnace. The preliminary coking of the coal eliminated the volatile hydrocarbons and concentrated the fixed carbon, which only is of value in the furnace. Heating the blast by means of the escaping gas, which has performed its function in the furnace, recovers a portion of its potential heat and returns it to the furnace at the point where, owing to the nature of the process, it is most effective in oxidizing the solid fuel. Concentration of the ores, whether natural or artificial, as well as the elimination of the moisture from the blast, reduces the heat-requirements and therefore the fuel-consumption.

According to the principles already known, it may be asserted, without any claim to the gift of prophecy, that there are to-day apparently no reasons, except commercial ones, why

iron should not be produced in the blast-furnace with 1,200 lb. or less of coke per ton. In the tables given with this paper, furnace E is a hypothetical one, having the same conditions as natural-blast furnace C, except that the temperature of the blast is increased to a point where it will be equivalent in results to those effected in D (dry blast). The temperature found to be sufficient is 569° C. (1,056° F.). It will be noted, moreover, that in Table III. the items of heat-requirements from 1 to 9, inclusive, are precisely the same in E as in the dry-blast furnace, item 9 being smaller for E than for C (natural blast), for the same reason as made it smaller with D—namely, less fuel, and consequently somewhat less slag. Item 10 (the decomposition of the moisture) would, of course, be the same for E as for C (natural blast). It is in items 12 and 14 that the significant changes in heat-requirements occur; the heat carried off in the gas being, for E, lower than for C (natural blast), and somewhat higher than for D (dry blast). The heat-conditions for radiation are similarly affected.

As compared with C (natural blast), it will be noted that there is a small direct saving in heat-requirements (item 9) of 11 units, or 0.2 per cent., and an indirect saving (items 12 and 14) of 330 units, or 8.6 per cent., to which is to be added the increased amount of heat carried in by the blast (item 17), 67 units, or 1.7 per cent., making a total saving of 408 units, or 10.5 per cent., in the heat to be developed from the fuel as per item 18, which gives 1,881 units for C and 1,473 for E; difference, 408 units, or 10.5 per cent.

As the heat in item 18 comes from carbon burned to CO in the fusion-zone, the fuel saved would be $\frac{408}{2473} \times \frac{2240}{0.88} =$ 241 pounds.

Had the stove-equipment for C been of ample capacity, the application of the dry-air blast would have shown a still more remarkable decrease in fuel-consumption. This is demonstrated in hypothical furnace E1, in which the conditions are the same as in D (dry blast), except that the temperature of the blast is assumed to be 1,200° F. (650° C.), a temperature easily and safely maintainable with fire-brick stoves.

It will be noted in Table III., item 11, that there is a difference of only 3 units in the total of direct heat-requirements

between D (dry blast) and E1, representing the same conditions, except that the temperature of the blast is 330° F. (185° C.) higher. This small difference is due to the decrease in fuel, and consequently in the amount of slag from the ash, as shown in Table II.

In the indirect heat-requirements (item 15, Table III.) a saving for E1 of 16 units over D, carried off in the gas, is due to the smaller quantity of gas; and that of 9 units in radiation (item 14) is due to the smaller amount of heat developed and, as a natural consequence, less radiation. This makes a total saving for E1 in heat-units required of 28. Adding to this the increased heat carried in by the blast (470 — 370 = 100 units), we have a total of smaller heat-requirement from the fuel of 128. This agrees with item 18 (1,454 — 1,326 = 128 units), which is heat developed by the combustion of C to CO; therefore, the equivalent in coke is $\frac{128}{2473} \times \frac{2240}{0.88} = 131$ lb. saved by E1 over D, by reason of the increased temperature of the blast. The corresponding saving of E1 over the natural-blast furnace C would be 421 + 131 = 553 lb. of coke.

Furnace F shows what might be expected with atmospheric temperature 75° F. (24° C.), other conditions being the same as with dry blast. As might be expected, the heat-requirement for fusion of slag (Table III., item 9) is, by reason of increased ash from the increased quantity of fuel required, greater than in either D, E, or E1, and exactly equals the requirement in C (natural blast), as it should do, because, as will be seen later, the fuel required is precisely the same.

Under indirect heat-requirements, item 12 (carried off in the gas) is greater than in D, E, or E1, but considerably less than in C. The temperature of the gas being the same as in D or E1, while the quantity is considerably greater, the heat-requirement is, of course, greater. As compared with C and E, the temperature of the gas from F is lower, and the increased quantity of gas more than balances the effect of this factor in the case of E, but is not sufficient to do so in the case of C.

It will be noticed that the requirement for radiation (Table III., item 14) varies directly and almost in exact ratio with the heat developed in the fusion-zone of C, D, E, and E1. This was

to be expected, since the heat developed in the reducing-zone is nearly the same in all these furnaces.

Furnace G shows what might have been expected if, in addition to drying the blast, the moisture had been expelled from the ore before charging into the furnace. The saving in heat-requirements is shown as follows in Table III.:

	Heat-Units.
Direct saving:	
Item 3, expulsion of moisture,	109
Item 9, fusion of slag,	3
Indirect saving:	
Item 12, carried off in gas,	15
Item 14, loss by radiation,	24
Total saving in heat-requirements,	151

From this total saving must be deducted the decreased heat carried in by the blast (item 17), 31, leaving a total net saving in heat-requirements of 120.

Since the saving in heat-requirements all comes from fuel burned to CO, the equivalent in coke would be $\frac{120}{2473} \times \frac{2249}{0.88} =$ 122 lb. of coke less than would be required per ton of iron by D.

The conditions of furnace H are: dry blast; moisture of ore and carbonic acid of limestone expelled by calcination before charging; temperature of blast, 1,200° F. (650° C.); temperature of gas, 375° F. (191° C.); radiation, 10 per cent. of total heat developed. All of these conditions are practicable at the present time to the degree stated, and some of them could be made still more favorable.

Comparing furnace H with C (natural blast), it will be seen, Table III., that under direct heat-requirements, those of the reduction of iron (item 1), and silicon (item 2), and the fusion of iron (item 8) are precisely the same. Since, under H, the moisture in the ore has been expelled, the only heat-requirement of this nature is 3 units (item 3) for the moisture contained in the coke. Item 4, for carbonic acid, drops out. Less fuel, and consequently less ash and slag, reduces item 9; and, of course, as we have seen, the dry blast makes less requirement for decomposition of the moisture in the blast (item 10). Altogether there is a total saving in direct heat-requirements of 421 units, or 10.91 per cent. of the total for C.

In indirect requirements there is a saving in heat carried off in gas (item 12) of 268 units, or 6.95 per cent. This is due to the large decrease in the quantity of gas (Table II., item 15), amounting to $\frac{5.990 - 2.903}{5.990} = 51.5$ per cent., which, in turn, is due to the decreased quantity of fuel and blast required, and to the lower temperature of the gas. In the item of radiation (Table III., item 14) there is a decrease of 294 units = 7.62 per cent. This is due to the decreased quantity of total heat supplied (item 23), and also to the lower temperature in the reducing-zone, as evidenced by the lower temperature of the escaping gas.

As to heat supplied in the fusion-zone, that carried in by the blast (item 17) for H is 4 units, or 0.10 per cent., less than in C (natural blast), notwithstanding the large increase in temperature. This is due to the fact that the decrease in quantity of blast overcomes the gain from increased temperature. This item deducted from the total decreased heat-requirements represents the total decrease of heat required to be supplied from the fuel.

Tabulating these results, we have:

Item No.	Units.		Per Cent.
11, saving in direct requirement, ...	421	=	10.91
12, saving in heat carried off by gas, ...	268	=	6.95
14, saving in radiation, ...	294	=	7.62
	983		25.48
17, deduct less heat carried in by the blast, . .	4	=	0.10
Total net saving of heat-requirements, ...	979	=	25.38

This all comes from fuel burned at the tuyeres, and is shown in item 18 (1881 — 902 = 979 units), being equivalent in coke to $\frac{979}{2473} \times \frac{2240}{0.88} = 1{,}009$ lb., saving in coke per ton of pig-iron as compared with natural blast, or a total of 1,138 lb. of coke per ton of iron.

The conditions of furnace I are precisely the same as those of II, except that the nitrogen has been eliminated from the blast. Comparing with H, it is seen (Table III.) that the heat-requirement for the expulsion of moisture (item 3) is increased 1 unit, due to the increased fuel required, and the requirement for the fusion of slag (item 9) is increased 6 units for the same

reason, while the requirement for decomposition of the moisture in the blast (item 10) is decreased 14 units, making a total net decrease in direct heat-requirements (item 11) of 7 units.

Under indirect heat-requirements, item 12 (carried off in gas) is reduced 62 units by reason of the decreased quantity of gas, and item 14 (radiation) is reduced 9 units by reason of the decreased quantity of heat supplied (item 23).

Of the heat supplied, there were 300 units less carried by the blast into I than into H (item 17, Table III.).

Tabulating these results, we have:

Comparison of H with I.

Item No.		Heat-Units.
17, less heat carried by the blast with H,		300
11, decrease in direct requirements,		7
		293
12, saving in heat carried off in gas,	62	
14, saving in radiation,	9	71
18, increased heat required (1,124 — 902),		222

This heat derived from fuel burned to CO at the tuyeres would be equivalent in coke to $\frac{222}{2473} \times \frac{2240}{0.88} = 229$ lb. more than in furnace H, or, in other words, I would require 1,367 lb. of coke per ton of iron, showing that the elimination of nitrogen from the blast, instead of reducing, would increase the fuel-consumption.

Furnace J, the Clarence furnace of Bell Brothers, shows strikingly how the use of lean ores affects the fuel-consumption. Table I. shows that in this furnace 2.4 tons of ore were required for 1 ton of iron, requiring 0.55 ton of limestone as flux, and producing from these materials, together with the ash from the coke (Table II., item 15 under J), 1.391 tons of slag as against 0.58 ton in the natural-blast furnace C, while requiring in direct heat for the one item of the fusion of slag (Table III., item 9) an increase of 770 — 290 = 480 heat units, equivalent to an increase in coke of $\frac{480}{2473} \times \frac{2240}{0.88} = 482$ lb. per ton of pig-iron.

Furnace K, Union furnace No. 1 of the Illinois Steel Co., shows the effect of using a rich ore, only 1.57 tons of ore (Table I., K, 5) and 0.27 ton of limestone (Table I., K, 6) being used and, together with the ash from the fuel, producing only 0.3 ton

of slag (Table II., K, 15) per ton of iron, thus saving in direct heat-requirement for the fusion of slag (Table III., item 9) $290 - 165 = 125$ units, equivalent in coke to $\frac{125}{2473} \times \frac{2240}{0.88} =$ 129 pounds.

Conclusions.

Upon careful consideration of the foregoing facts and figures, it will be evident that there are no more startling economies in fuel-consumption by the iron blast-furnace to be achieved, further improvement in the air-blast elimination of the nitrogen having been shown to be a detriment instead of a benefit.

With conditions already practically reducing the coke to about 1,200 lb. per ton of iron, the saving to be effected by carrying a blast-temperature above 1,200° F., say at 1,600° F., would be considerable; but, since reserve heat must be carried somewhere for emergencies, it is doubtful whether it would be advisable in practice to carry regularly more than 1,200° F.

Improvements in practice will no doubt enable us to reduce the moisture of the blast somewhat below 1.75 grains per cubic foot, and thus effect some additional saving in fuel. But more important future progress in the economy of blast-furnace fuel must be the result of a more careful preparation of the ore, limestone, and fuel, eliminating from these materials the moisture and volatile matter, and reducing the slag-making elements. Of course, the cost of such preparation will have to be weighed against the resultant saving of fuel in the furnace-process itself, and the scientific furnace-manager will find the problem with which he has to deal not less difficult than were the cruder problems of a more ignorant practice. In fact, as we all know, the result of scientific progress in our art is to make our responsibility greater and our task more complex. Pioneers like Mr. Gayley certainly confer great benefits upon capitalists, workingmen, and the public at large; but it cannot be denied that they worry their professional colleagues!

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Vanadium-Deposits in Peru.

Discussion of the paper of D. Foster Hewett, *Bulletin* No. 27, March, 1909, pp. 291 to 316.

JAMES F. KEMP, New York, N. Y.:—Mr. Hewett's paper is one of exceptional interest, because it not only adds an important contribution regarding one of the rarer, valuable elements, but also because it introduces compounds hitherto unknown and in associations no less novel. While we have occasionally discovered and recorded asphaltite in metalliferous deposits, for example in the Joplin and Granby districts, we have not seen a metallic ore in a deposit essentially asphaltite. The source and method of introduction afford a subject well worth serious reflection.

Vanadium was first discovered and recognized as a distinct element in the slags obtained in smelting the titaniferous magnetites at Taberg, Sweden. For a long time, therefore, we looked upon the titaniferous magnetites as its home. Analyses proved, years ago, that it was present in the ores of this type in New Jersey and in the Adirondacks. It was quite rarely determined, but when looked for it was, I think, invariably found, although the amount seldom exceeded 0.5 per cent. of V_2O_5. In just what combination it exists in the titaniferous ores is uncertain. One might imagine V_2O_3 replacing some of the Fe_2O_3 of magnetite; or, since in the lead series vanadates and phosphates are closely akin, one might wonder if a lime-vanadate could take the place of apatite. Yet no lime-vanadate has been found in nature, and apatite itself is usually rare in the titaniferous ores. When percentage-curves are plotted for a series of analyses of the ores, the line of vanadic oxide shows a curious sympathetic behavior with the line of chromic oxide;[1] but the data are somewhat limited, and no compound has suggested itself which throws light on the matter.

Dr. W. F. Hillebrand has discussed the occurrence and dis-

[1] *Nineteenth Annual Report, U. S. Geological Survey,* Part III., Pl. LV., opp. p. 394 (1897–98).

tribution of vanadium in nature, and concludes that it favors the moderately basic eruptives.[2] In these it may possibly replace the Fe_2O_3 of the silicates, but it does not favor richly magnesian rocks, presumably because there are few sesqui-bases in them to replace. H. S. Washington has also given a brief summary of its distribution in nature and reaches the same conclusion.[3]

For many years vanadium has been known as a constituent of the so-called coals of Peru and Argentina. In Hillebrand's paper, a contribution by A. A. Hayes is cited in which Dr. Hayes, writing in 1875, quotes Thorpe's *Dictionary of Chemistry*. The last named, obviously of still earlier date, states that a coal from Peru contained 0.45 per cent. of V_2O_5, and that two samples of ash gave 38.5 and 38 per cent. As a constituent of Argentine coals, it is mentioned in the *U. S. Consular Report for* 1894, p. 176, and three or four years later I received a letter from R. S. McCaffery, at the time chemist at the smelter in Casapalca, Peru, announcing that he had found notable percentages in the ashes of a coal used at the smelter. It was apparently supposed at this time that the so-called coals were of the usual sedimentary types, and it was always a puzzle as to the source of the vanadium. Mr. Hewett has now shown that they are all asphaltites.

Asphaltite-veins are generally believed to have been formed by the entrance of a heavy petroleum, with an asphalt base, into a fissure. Naturally, we would ascribe to the petroleum a source in some oil-pool below. This explanation would suggest itself for the case described by Mr. Hewett. With all the reservation proper to one who has not seen the district, and with full appreciation of the careful work done by Mr. Hewett, who rather favors a derivation of the vanadium by segregation from the neighboring sedimentaries, the uprising petroleum bringing with it the patronite appeals to me rather strongly. This method of vein-filling was trusted for the grahamite-vein of West Virginia, and an oil-pool was subsequently found at a depth of 1,500 to 1,600 ft. by drilling near it.[4] It would be of

[2] W. F. Hillebrand, Distribution and Quantitative Occurrence of Vanadium and Molybdenum in Rocks of the United States, *American Journal of Science*, Fourth Series, vol. vi., No. 33, p. 209 (Sept., 1898).

[3] *Trans.*, xxxix., 756 (1909).

[4] I. C. White, *Bulletin of the Geological Society of America*, vol. x., p. 278 (1898).

much interest to know if vanadium sulphide is soluble in these heavy, asphaltic oils, and especially in the sulphurous varieties. Oils must traverse extended sections of rock before gathering in quantity and may thus pick up vanadium. We may note that the moderately basic rocks, dolerite and diabase, the former in a laccolith, the latter in a dike, are both reported near the asphaltite vein.

Asphaltite is not unknown in deposits of the metallic ores, as has been remarked above. It may be that when graphite appears, as it does in the great sulphide-veins at Ducktown, Tenn., it represents some original hydrocarbon. But a heavy oil as a solvent of a metallic or semi-metallic sulphide, and as the vehicle of its introduction, is something new.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Genesis of the Lake Valley, New Mexico, Silver-Deposits.

Discussion of the paper of Charles R. Keyes, *Trans.*, xxxix., 139 to 169.

WILLIAM M. COURTIS, Detroit, Mich. (communication to the Secretary*):—I have a few items to add to the history of the Lake Valley mines.

In December, 1879, I was sent to the Bassic mine of Colorado and then to Lone mountain, near Silver City, N. M., and to various copper-camps in Arizona. I staged from Glorietta, N. M., to Tucson, Ariz., stopping at Silver City to see the Cosette mine. This mine led to the opening of the Lake Valley mines, and is a deposit of similar formation.

New Year's day, 1880, found me crossing the Rio Grande above the Jornado del Muerte. At Aleman, Victorio's band crossed four hours ahead of us. We picked up four wounded soldiers to take to Fort Bayard, for during the day the colored troops under General Hatch had been driving the Indians back through the San Andreas mountains. The traveler of to-day little realizes the dangers of these trips, when over very rough roads we made about 100 miles in a day and a night.

I subsequently made this trip six times, up to 1882, and each time I missed an Indian attack by from one to four days. The day I arrived at Silver City news had come of the massacre of 75 women and children at the San Francisco ranch. The fighting-men of Silver City had gone to rescue others. I was the only passenger from Socorro, N. M., until we picked up the soldiers. The next day we came suddenly on 5 of Victorio's band, but we had picked up several miners, so our stage bristled with guns, and we were let alone.

At or near my stone quarry, 6 miles from Socorro, 9 men were killed by the band that, a day or two after, killed George Daly, Lieutenant White, and all their men. During this raid our miners and women at the mine slept in the tunnel. At the Cosette mine I had 35 men at work. These Indian raids had

* Received Feb. 23, 1909.

killed off the freighters, so for a time the only supplies we had were beef, oatmeal, honey, and a few canned tomatoes.

The Cosette mine was on a trachyte dike, breaking through the limestones. The vein carried lead carbonates and horn-silver, as rich as 1 oz. in 6 of ore, averaging $100 in value. Some $20,000 worth had been taken out in the two months during which I was in New York, and I was sent back immediately to test this mine. A very rich semi-opal and hyalite showed that the deposit had been made by hot water. The vein dipped quite steeply for a short distance, then ran nearly flat, and pitched again. The flat parts carried the rich horn-silver, 90 lb. being our largest mass, of which one-sixth was silver. Immediately after this find we broke into a cave of greater depth than we could measure and very hot. It was not over 18 in. wide, and was lined with ore that ran in a small testing-mill from $35 to $52 per ton. For a mill, I had found a flowing stream, by noticing that the prairie-dogs had dug up wet dirt in this desert. It was only 16 ft. down to water. The mine would have paid for a mill, but the parties sold it, as the drop in the ore discouraged them, though for a working-expense of $13,000, and hauling the ore 5 miles, we had taken out more than $8,000 in the testing.

While working here the Indians had raided the country about Lake valley and killed some of the men. The McEverts ranch covered the mines then known only as possible prospects. One of my men said that Mrs. McEverts wanted to sell the ranch for $5,000, and that there were good signs of ore like ours. I had $5,000 to invest for parties, but the Indians were too close just then to go there, and I would not take it without seeing it. John A. Miller, sutler at Fort Bayard, took a guard of soldiers and looked at the ranch, making a bargain to purchase it at about the price offered to me, thinking of it only as a hay-ranch. Seeing this very rich ore I had found, he set some men to work on similar outcroppings. I assayed the samples his men found. At first these did not run over $42, but in a few days one sample ran about $10,000, and Mr. Miller made me an offer of a half interest if I would put up a small smelter, to cost $20,000. I took the matter up with Capt. W. H. Stevens, of Detroit, owning the Iron-Silver mines of Leadville, who did not accept the proposition.

Shortly after, Mr. Miller told me that he had sold part for $125,000, and the purchasers made me an offer to come back and put up the works, but those with whom I had an engagement would not release me. Mr. Miller told me he received $350,000 for all his interests in this property. I was idle some months under salary at Detroit waiting orders, until sent to Socorro to open up the Torrence mine. I understood that R. Bunssen, of Leadville, had built a smelter at Silver lake that was not successful on the class of ore they opened into. George Daly and the other officers I met gave me pieces of what they called "Jackson's baby," a solid mass of horn-silver, found, as I think, in the "Bridal Chamber." I have some pieces now.

So many prominent men were killed by the Indians in 1882 that no one would look at any New Mexican property for years, but some day much gold will be obtained from neglected mines. I opened great bodies of $10 gold-ore, but as the expense was $12 per ton we could do nothing with it. I had found out in 1881 that our ore would extract to 90 per cent., with a 10-per cent. solution of cyanide, but with cyanide at $2 per pound it did not look attractive for a $10 ore.

On the trip early in 1880 I was given a 10-day option for $80,000 on 22 claims at Bisbee, including the Copper Queen. The owners had been driven in by the Indians, and brought with them many burro-loads of beautiful ore. I sent four 4-lb. sacks to a Calumet & Hecla stockholder in Boston, but the over-wise mining-man of that company, to whom the matter was referred, whose experience had been limited to the Lake Superior region, said there were no copper-deposits in Arizona that would go down, so I received no aid. Many years after, when I was looking at the ore-stopes above the sixth level, with ore running about 30 per cent. of copper, I wished every young mining engineer could profit by this lesson, and be careful not to make too wide a generalization from a limited experience. The work I laid out on the belief that the copper would go down in Arizona made my clients a very satisfactory return.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Ozark Lead- and Zinc-Deposits; Their Genesis, Localization, and Migration.

Discussion of the paper of C. R. Keyes, presented at the Chattanooga meeting, October, 1908, *Bulletin No.* 26, February, 1909, pp. 119 to 166.

E. R. Buckley, Flat River, Mo. (communication to the Secretary*):—Some statements in the paper of Mr. Keyes relative to the nature and formation of the Ozark lead- and zinc-deposits seem to me erroneous and misleading, and I respectfully present the following criticisms, placing Mr. Keyes's original statements in quotations.

Referring to the Ozark dome: (P. 127.) "It is a region that has been repeatedly upraised and planed off, until in the middle portion the oldest known rocks only are exposed." Does he mean that the St. Francois mountains, which have always been considered as lying on the eastern flank of the Ozark dome, are the middle of the dome? Or does he mean that the Cambrian rocks of the generally-accepted middle are the oldest known rocks? Certainly not the latter, and if the former, his idea of the middle of the dome is not in accord with that of others who are familiar with the region.

(P. 138.) ". . . the ore-deposits are definitely associated with, or localized by, geologic structures of some kind that produce in effect basins in which the underground waters are impounded, or their flow retarded." This statement does not appear to be altogether consistent with the facts, since the ore-bodies of the Joplin district, as the rule, occur in those parts of the Mississippian formation which have a tendency to accelerate the flow of ground-water rather than retard it. I might name the open, brecciated ground near Joplin, Webb City, and Oronogo as especially noteworthy illustrations. The ore-bodies, everywhere, occur in positions where there must have been a free and copious circulation of dilute lead- and zinc-solutions in the presence of reducing constituents.

The inequalities in the distribution of the ore-deposits are

* Received Mar. 11, 1909.

accounted for in the following manner: (P. 135.) "The effects of crustal deformation have been unequal. In consequence of this, the accumulation of the ore-materials is also more or less unevenly disposed." Thus by implication he rejects all agencies such as brecciated or "open" ground, the presence of organic material, etc., as important in the segregation of the ores. Neither does he consider the possibility of an irregular distribution of the metals in the various systems of underground circulation.

(P. 131.) . . . "the recognition of ores of the first circulation and of a subsequent second concentration need not enter into consideration." What is meant by "first circulation"? Does he refer to "primary concentration," as the expression is generally used?

(P. 134.) "Areally, the lead- and zinc-deposits of the Missouri-Arkansas region are mainly distributed in a belt of greater or less width which borders the basal margin of the Ozark dome and completely encircles it." In referring to the central Missouri district: (P. 139.) "While many minor ore-bodies still remain in this upper zone, the extensive ore-bodies of the central district must be sought at deeper levels than in other parts of the Ozark region—at, or below, the present deep-lying permanent water-level." This "central district" actually lies near the middle of the dome referred to, and the inference one would draw from this statement is that extensive ore-bodies occur here as well as "in a belt" bordering the basal margin of the dome.

Again referring to this district: (P. 139.) "Whatever ore-bodies once existed in this upper zone have been largely removed. This is shown to some extent by the myriads of caverns throughout the region." Because this is a cavernous region it is not to be assumed that it was once a richly-mineralized region.

Where will one find facts warranting the statement that "all of the known mines [in the central Missouri district] are located in certain straight and narrow belts, which are near and parallel to the axes of shallow synclines"? And that "these synclines pitch radially from the center of the Ozark uplift"? (P. 130.) Keyes gives none, and since most of the recent work in central Missouri has been done by the Missouri Bureau of Geology and

Mines without developing any such relation, his generalization is open to serious doubt.

(P. 165.) "Thus, as the Coal Measures margin retreated down the slope the ore-belt also continually migrated in the same direction, until its present position was reached." This statement hardly appears consistent with those relative to the occurrence of "extensive ore-bodies" "at deeper levels" in the central district.

Keyes lays very great stress upon the occurrence of a broad syncline trending westward through the so-called Joplin district, combined with SE–NW. synclines, of which he says there are no less than four, conforming to major ore-runs near Joplin, Carthage, Galena, and Cartersville. He simply makes the statement that these synclines exist. There is no proof of it in his paper, and I know of none elsewhere available.

He speaks of the intimate relations existing between the Coal Measures remnants and the ore-deposits of the southwest Missouri district as "accidental associations." (P. 151.) If this is true, the "accidental associations" are so numerous as to justify some hesitancy in accepting the statement.

(P. 165.) "The genetic relationship of ore-runs to buried relief-features at the base of the Coal Measures does not obtain." This statement is not accompanied by proof, and its truth should be demonstrated before being accepted.

Although one would scarcely infer as much from Keyes's paper, the demonstration that Bain lacked proof of the existence of major faulting in the Joplin district was first published in the Geology of the Granby Area.[1] Keyes evidently concludes, as others have done, that because faulting is not associated with the ore-deposits of the Joplin district, it is safe to conclude that this structure has no relation to the ore-deposits of other districts in this region. As a matter of fact, the ore-deposits of southeast Missouri are closely associated with faults, which have evidently played an important part in localizing the deposits of galena. The detailed facts upon which this statement is made are given in a work now in press.[2]

[1] *Report of the Missouri Bureau of Geology and Mines*, Second Series, vol. iv. (1905).

[2] *Report of the Missouri Bureau of Geology and Mines*, Second Series, vol. ix. (1909).

Keyes refers to the ore-bodies of the southeast Missouri lead-district: (P. 130.) "In the southeast Missouri district the unconformity-plane marking the base of the Cambrian terranes seems to have largely controlled the localization of the great bodies of disseminated ores." . . . (P. 146.) "Many of the principal ore-bodies now lie near the bottoms of these ancient drainage-troughs. It seems probable that most, if not all, of the ore-deposits of the district will be eventually found to have some direct connection with the courses of the old troughs. Observations bearing upon this very point were published nearly 15 years ago, and since that time practical tests have fully confirmed the original working hypothesis." . . . (P. 160.) "In the southeast Missouri lead-district the localization of the ore-bodies appears to be chiefly influenced by the character of the pre-Cambrian channel-ways corraded out of the still older granites." . . . (P. 164.) "In the southeastern district the general conditions are also like they are in the southwestern area. Corresponding to the warped surface is the uneven erosion-plane of unconformity at the base of the Cambrian rocks. Instead of porous zones caused by brecciated cherts and limestones are the porous dolomites."

I do not enter into a discussion of the ore-deposits of the southeast Missouri district, since I have already covered this subject in detail.[3] If Keyes were in any degree familiar with developments during the last 10 years, he would hardly make the statements above quoted. Winslow pointed out long ago that the ore-bodies of the Bonne Terre and Flat River areas lie in a pitching trough, but my investigations have not disclosed any relation between the minor troughs and the localization of the ore-bodies. The position of the ore-bodies in this pitching trough depends upon several other factors, including faulting, jointing, and sedimentation.

In recapitulating: (P. 165.) "The geographic distribution of the main ore-deposits is circumscribed, the belt in which they are confined forming a continuous circle around the base of the great dome." This has yet to be proven. Thus far there have actually been found only two "main ore-deposit" districts, one in the southwestern part, including Jasper, Newton, Lawrence,

[3] *Report of the Missouri Bureau of Geology and Mines*, Second Series, vol. ix. (1909).

and Greene counties, and the other in the southeastern part of the State, including St. Francois, Washington, Franklin, and Madison counties. The remainder of this "continuous belt" is dotted with prospects, the combined output from which will not equal that of a single mine in St. Francois county. Of the so-called "continuous belt," 75 per cent. is territory which has not yet been proved to have noteworthy deposits of either lead or zinc. Such deposits as occur over this portion of the belt are of little, if any, greater importance than hundreds of others scattered irregularly over the entire Ozark region of Missouri.

(P. 165.) "Primary source of the ore-materials is a factor the importance of which has been very greatly overestimated, and is of no significance in practical mining-operations." There are many pre-Cambrian erosion-basins in southeast Missouri similar in all respects to that in which occur the disseminated-lead deposits of St. Francois county. They are in all respects, except in the matter of faulting and source of ground-water, under the same conditions, but diamond-drilling seems to indicate that they do not contain deposits of lead. How would Keyes account for this condition? If he were operating in this district I am sure he would regard faulting, character of sedimentation, and primary source of the ore-materials as of primary importance.

(P. 156.) "The diffused metallic content of the Ozark rocks cannot, therefore, be regarded as derived in any way from the old crystalline basal complex now exposed in the region." If it did not come from the removed portion of this pre-Cambrian complex, where did it come from? From distant areas? Perhaps, in part; but it must be remembered that the rocks of the Palæozoic succession in Missouri are essentially near-shore deposits, and it must be conceded that distant continental areas contributed only subordinate amounts of material to their mass.

Keyes uses a remarkable series of illustrations. To my knowledge they do not represent a single concrete condition in the lead- and zinc-districts of Missouri. Where is the "syncline on the Osage river," Fig. 3; "the unconformity-trough at Doe Run," Fig. 5; the "warp-sags" of Fig. 6; the "silted-up cavern," near Aurora, Fig. 7? Where in the disseminated-lead district do the ores occur as shown in Fig. 10? What is the basis upon which the author worked out the conditions

shown in Fig. 18? Fig. 8 is referred to as "a very detailed picture of the geologic structure in a direction parallel to the margin of the Ozark dome and transverse to the distinct minor flexing." If this is a detailed section, I should like to see a generalized section. Some of the men who have been studying the ore-deposits of the Ozark region for a number of years would be glad to know the exact localities from which these drawings were made. Take Fig. 10, for example; any one who has ever been in the disseminated-lead district knows that the ore now being mined occurs above the Lamotte sandstone, and that only where the sandstone is absent does it rest upon the granite. Yet Keyes represents it as occurring at the base of the Lamotte sandstone.

The man engaged in mining pursuits is prone to criticise the geologist for publishing theoretical dissertations which have little foundation in fact. Mr. Keyes's paper contains practically no facts relative to the lead- and zinc-deposits of the Ozark region, and without them the theoretical discussion is only of value in so far as the reader may believe the unqualified statements. It is easy enough to deny the existence of certain conditions, stratigraphic and structural, but it is another matter to prove this denial. Likewise, it is easy to assert the existence of certain phenomena, but an entirely different proposition to prove their existence.

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Bulletin of the American Institute of Mining Engineers.

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EDITORIAL OFFICE AT 29 WEST 39TH STREET, NEW YORK, N. Y.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.

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SECTION I.—INSTITUTE ANNOUNCEMENTS.

This section contains announcements of general interest to the members of the Institute, but not always of sufficient permanent value to warrant republication in the volumes of the *Transactions*.

SECTION II.—TECHNICAL PAPERS AND DISCUSSIONS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

A detailed list of the papers contained in this section is given in the Table of Contents. They have been so printed and arranged (blank pages being left when necessary) that they can be separately removed for classified filing, or other indedendent use.

A small stock of separate pamphlets, duplicating the technical papers given in Section II. of this Bulletin, is reserved for those who desire extra copies of any single paper.

Comments or criticisms upon all papers given in this section, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

All communications concerning the contents of this Bulletin should be addressed to Dr. Joseph Struthers, Assistant Secretary and Editor, 29 W. 39th St., New York, N. Y. (Telephone number 4600 Bryant).

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

INSTITUTE ANNOUNCEMENTS.

Spokane Meeting and Excursions.

The Institute meeting at Spokane, and the preceding or subsequent excursions and visits to the Yellowstone National Park, Butte, Anaconda, the Cœur d'Alene, Seattle, Tacoma, Portland, the Columbia river from Portland to the Dalles, Salt Lake City and vicinity, Glenwood Springs, Colo., the Cañons of the Grand and Eagle rivers, Leadville, the Royal Gorge of the Arkansas, Pueblo and Colorado Springs, constituted one of the most instructive and delightful episodes in the history of the Institute. The journey of the party of visiting members and guests deserves to be ranked for professional interest and esthetic and social pleasure with the memorable former trips to California, British Columbia and Alaska, Canada, Mexico, Great Britain and Germany. A report of the proceedings and an account of the excursions and entertainments will be published in the *Bulletin* for December.

Meetings of Other Societies.

Canadian Mining Institute.—A brilliant company of representative mining-men gathered on Friday, October 22, at a complimentary banquet in the Canada Club, Montreal, tendered to J. Stevenson Brown, upon his retirement from the office of Treasurer of the Canadian Mining Institute, which he had filled to universal satisfaction for ten years. The chair was occupied by Major R. G. Leckie, of Sudbury, Ont., the Dean of the Institute, who, in proposing the health of the guest of the evening, associated his name with that of the late Secretary, B. T. A. Bell, the founder of the organization, and, after praising the devoted services of the Treasurer, presented him, in the name of the members of the Institute, with a handsome silver service. Speeches were made by leading representatives of mining from Nova Scotia, Ontario, British Columbia, and the United States.

International Congress for Mining, Metallurgy, Applied Mechanics, and Practical Geology, Düsseldorf, 1910.

As already announced in *Bulletin No.* 32, August, 1909, the Congress will meet at Düsseldorf during the last week in June, 1910. Extensive preparations are in progress, including visits to technical institutions and industrial establishments, excursions to localities of geologic interest, etc., which will have a direct bearing on the papers and addresses presented at the meeting.

The following information, dated October, 1909, has been received from the Secretary of the Congress:

Membership:

There will be two grades of membership of the Congress, as follows:

1. Members, who are entitled to become patrons of the Congress by payment to the funds of a contribution of not less than 100 marks (£5).

2. Members, who pay a subscription of 20 marks (£1).

The first-named class of members or patrons will be entitled to receive the printed reports of all proceedings of the Congress and of all its sections. Members of the second class, or ordinary members, will receive the reports of that section only in which they enroll themselves. The proceedings of any one of the other sections are obtainable at an additional charge of 5 marks (5s.) in each case.

Meetings and Excursions:

The work of the Congress will be performed:

1. In General Meetings, at which various papers of general interest will be presented.

2. In sectional Meetings, for the purpose of discussing important problems relating to Mining, Metallurgy, Applied Mechanics, and Practical Geology.

3. By making visits to scientific Institutions and industrial undertakings, etc., and by excursions to districts of geological interest.

For the purpose of entertaining ladies accompanying members to the Congress, a Ladies' Committee will be formed, which will endeavor in every way to render the stay of the lady visitors in Düsseldorf as agreeable and enjoyable as possible.

Provisional Program:

Section I: Mining.

1. Shaft-sinking, with special reference to the cement processes, freezing processes, and tubbing of shafts at great depths. The lining of shafts with concrete and reinforced concrete.
2. Methods of working, mine supports, with special reference to hydraulic packing, the use of reinforced concrete, the preservation of timber, and lighting of collieries.
3. Winding and haulage, with special reference to winding ropes, safety-catches and appliances, underground haulage, and haulage from the working faces.

4. Mine drainage.
5. Risks arising from fire-damp, coal-dust, and underground fires, and the methods of combating these.
6. The mechanical preparation of coal and ore, the recovery of bye-products, briquetting, and the utilization of low-grade fuels.
7. Recent practice in mine surveying.
8. Statistics.
9. Sanitation and hygiene.

Section II: Metallurgy.

A. Production of pig iron.

1. Coking.
   (a) Ovens.
   (b) Mechanical appliances.
   (c) Recovery of bye-products.
2. Ore supply.
   (a) Recent discoveries of ore.
   (b) Recent development and prospects of the ore briquetting processes.
3. Metallurgy of the blast-furnace process.
   (a) Influence of foreign substances.
   (b) Composition of slag.
4. Blast-furnace working.
   (a) Ore conveyance, storage, and charging.
   (b) Gas-washing and purification of the waste water.
   (c) Dry air-blast.
   (d) Casting machines and mixers.
5. Utilization of waste products.
   (a) Gases.
   (b) Dust from blast-furnace gases.
   (c) Slag (for hydraulic packing, cement, stone, concrete).

B. Production of malleable iron.

1. Advances in the methods of the metallurgical treatment of iron and steel.
   (a) Air-blast refining processes.
   (b) Open-hearth refining processes.
   (c) Processes for the production of electro-steel.
2. Production and treatment of special alloys of steel.

C. Iron and steel manufacture.

1. Improvements in the methods of casting iron and steel.
2. Further treatment of malleable iron.
   (a) Forging and pressing.
   (b) Rolling.
   (c) Fitting.
   (d) Development of the welding processes.
3. The driving of rolling-mills considered technically and economically (steam, gas, electricity).

D. Testing of iron and other metals.

1. Chemical testing.
2. Mechanical testing.

3. Metallography and microscopy of metals.

E. Economics of the iron industry.

1. Iron trade statistics.
2. Labor conditions and labor supply.
3. Patent rights.

F. Advances in the metallurgy of non-ferrous metals.

Section III: Applied Mechanics.

1. History of machine construction for mining and metallurgical purposes.
2. Steam raising.
3. Central electric power stations.
   (a) Reciprocating engines (steam, gas).
   (b) Turbo-engines.
4. Central condensing plant.
5. Winding engines.
   (a) Steam winders.
   (b) Electric winders.
   (c) Safety and signalling apparatus.
6. Pumping.
7. Fans and air-compressors.
8. Blowing engines for blast-furnaces and steel works.
   (a) Reciprocating blowers.
   (b) Turbo-blowers.
9. Methods of driving rolling-mills.
10. Rolling-mills and accessories.
11. Conveyors for mining and smelting works.
    (a) For materials in bulk (coal or coke).
    (b) Special cranes and ladle-cars.
    (c) Loading and unloading apparatus.

Section IV: Practical Geology.

1. Importance of practical geology in science and political economy.
2. Stratigraphy and genesis of the available mineral deposits. Calculations of their yearly output and resources.
3. Seismology, terrestrial magnetism, and terrestrial heat.
4. Questions relating to hydrology.
5. The utilization of natural sources of water power. Barrages.

The Committee on Organization request that applications for membership, accompanied by a remittance through the Post Office of the amount of the subscription, be made as soon as possible, in any case not later than Mar. 1, 1910. The remittance should be made payable to the "Stahlwerksverband A.-G. Düsseldorf, Kongress 1910."

Inquiries concerning membership in the Congress, announcement of proposed papers, length and time of presentation, suggestions, etc., may be addressed to Dr. E. Schrödter, Secretary of the Committee of the Congress, Düsseldorf, Jacobistrasse 3/5, Prussia.

Office Facilities for Visiting Members.

A separate room in the suite occupied by the American Institute of Mining Engineers on the ninth floor of the United Engineering Society Building, has been equipped with furniture and telephone extension for the temporary use of members of the Institute or of sister societies, or visitors suitably accredited.

Members of the Institute visiting New York for a short time, who need office facilities during their stay, or members residing in the city who need temporary office accommodation, can arrange to have set apart for their exclusive use a room, equipped with office furniture, telephone, etc., in the suite of the Institute. It is not the intention to give possession of the room to any individual for an indefinite time, but to offer to members of the Institute an opportunity to acquire a well-located, well-equipped business headquarters to carry on transactions which would not warrant the establishment of a permanent office. The room devoted to this purpose is entirely separate from the reception- and writing-rooms for the general use of the members. A small fee will be required for the use of the facilities furnished. For the conditions of this privilege, inquiry should be made at the office of the Secretary of the Institute.

How to Use the "Transactions" of the Institute.

Buy a copy of the Complete Analytical and Alphabetical Index of Volumes I. to XXXV., inclusive.

If you own a full set of the *Transactions*, this Index will make the whole of it instantaneously available without detailed research into each volume separately.

If you do not own such a set, this Index will be even more valuable, for it will show you what particular papers you need to know more about, and perhaps to study. Thus, any person possessing this Index can ascertain at once what has been published in the *Transactions* on a given question, and can learn, by writing to the Secretary, what is its nature, whether it is still to be had in pamphlet form, where it can be consulted in a public library, at what cost it can be copied by hand, etc., etc.

In short, to those who own complete sets of the *Transactions*, this Index will be a great convenience; but to those who do not, it will be a professional necessity.

This volume is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. It has not been stereotyped, and the edition is limited to 1,600 copies. The price of the volume, bound in cloth, is $5, and bound in half-morocco to match the *Transactions*, $6. The delivery charges will be paid by the Institute on receipt of the above price.

Hydrographic Chart.

Owing to the great value to hydrographers of the chart contained in the paper, A Graphic Solution of Kutter's Formula, by L. I. Hewes and Joseph W. Roe (*Bulletin No.* 29, May, 1909, p. 454), a special edition for office or field use has been printed on durable cloth. Copies of this separate chart may be obtained, at a cost of 50 cents each, on application to the office of the Secretary of the Institute.

Special Notice.

Attention is respectfully requested to the announcement on pages 12 and 13 of the advertising section of this *Bulletin* concerning certain books and periodicals needed to complete sets in the Library.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

RULES.

For the protection and convenience of members, the following rules have been adopted:

The Secretary of each Society will, upon application, issue to any member of his Society in good standing a personal, non-transferable card, entitling him to the use of the Libraries in the alcoves of the Reading-Room.

This card, as well as any card of introduction given to a non-member, must be signed by the person receiving it, and surrendered at the desk at the time of its presentation. At every visit he must identify himself by signing his name in the registry.

Strangers who desire to enjoy the privilege of entering the alcoves are requested to present either letters of introduction from members, or cards, such as will be furnished upon application by the Secretary of each Society. The first two alcoves are free to all; and admission to the inside alcoves is given upon proper introduction.

The above rules apply to all persons except officers of the three Societies, personally known as such to the librarians.

The librarians are not permitted to lend to any person any catalogued pamphlet or volume, unless authorized in writing so to do by the Secretary or Chairman of the Library Committee of the Society to which the pamphlet or volume belongs.

Any person discovering a mutilation or defect in any book of the libraries is requested to report it to the librarian on duty.

Library Additions.

From Oct. 1 to Nov. 1, 1909.

ACTION OF AIR AND STEAM ON PURE IRON. By J. N. Friend. (Advance paper, Iron and Steel Institute, 1909.) (Exchange.)

AMERICAN RAILWAY ASSOCIATION. Proceedings. Vol. IV. New York, 1906. (Gift of American Railway Association.)

ANNUAIRE STATISTIQUE DE LA BELGIQUE. Vol. 39, 1908. Bruxelles, 1909. (Exchange.)

ARMSTRONG COLLEGE. Calendar, 1909–1910. London, 1909. (Gift.)

ARTIFICIAL MAGNETIC OXIDE OF IRON. By F. J. R. Carulla. (Advance paper, Iron and Steel Institute, 1909.) (Exchange.)

BIBLIOGRAPHIE DER DEUTSCHEN ZEITSCHRIFTEN LITERATUR. Vol. XXIII. A. Leipzig, 1909. (Purchase.)

CARTE GÉOLOGIQUE INTERNATIONALE DE L'EUROPE. Livraison VI. Berlin, 1909. (Purchase.)

CIVIL ENGINEER'S POCKET-BOOK. 19th edition. By J. C. Trautwine. New York, J. Wiley & Sons, 1909. (Gift of Editors.)

[SECRETARY'S NOTE.—The words "By John C. Trautwine, Civil Engineer, Revised by John C. Trautwine, Jr., and John C. Trautwine, 3d, Civil Engineers, 19th Edition, 100th Thousand," constitute in themselves a review of this book; and if anything more were needed it is supplied by the bold but not unwarranted declaration of the publishers that "no pains have been spared to maintain the position of this as the foremost Civil Engineer's Pocket-Book, not only in the United States, but in the English language." For this edition the book has been extensively revised, and about 300 pages of new material (equivalent to about 600 pages of standard text-books) have been added. The most important of this new material is the series of articles on Concrete (plain and reinforced), including Cement, Sand, and Mortar; but there are also up-to-date chapters on Strength of Materials, Columns, Specifications for steel and iron, etc., Trigonometric Functions, Hydraulics, etc., reflecting the progress of engineering science and its increased demands.—R. W. R.]

CLAY PRODUCTS OF THE UNITED STATES, 1908. (Sheet.) Washington, U. S. Government, 1909. (Exchange.)

COLLOID MATTER OF CLAY AND ITS MEASUREMENTS. (Bulletin No. 388, U. S. Geological Survey.) By H. E. Ashley. Washington, U. S. Government, 1909. (Exchange.)

COMITÉ DES FORGES DE FRANCE. Annuaire, 1909–1910. Paris, 1909. (Exchange.)

COMPARATIVE STATISTICS OF LEAD, COPPER, SPELTER, TIN, ALUMINIUM, NICKEL, QUICKSILVER, AND SILVER. Compiled by the Metallgesellschaft, the metallurgischen Gesellschaft A. G., and the Berg und Metallbank Aktiengesellschaft. 15th Annual Issue, 1899–1908. Frankfort-on-the-Main, 1909. (Gift of Metallurgical Company of America.)

CONCRETE. By J. C. Trautwine, Jr., and J. C. Trautwine, 3d. New York, J. Wiley & Sons, 1909. (Gift of Authors.)

This manual is a reprint of the chapters on Cement, Mortar, and Concrete of the Civil Engineer's Pocket-Book mentioned above.

CONSTITUTION OF CARBON-TUNGSTEN STEELS. By T. Swinden. (Advance paper, Iron and Steel Institute, 1909.) (Exchange.)

CORROSION OF IRON. (Advance paper, Iron and Steel Institute, 1909. (Exchange.)

CUERPO DE INGENIEROS DE MINAS DEL PERÚ. Boletin. Nos. 70–74. Lima, 1909. (Exchange.)

DETERMINATION OF THE ECONOMY OF REVERSING ROLLING-MILLS. By C. A. Ablett. (Advance paper, Iron and Steel Institute, 1909. (Exchange.)

EFFECT OF OXYGEN IN COAL. (Bulletin No. 382, U. S. Geological Survey.) By David White. Washington, U. S. Government, 1909. (Exchange.)

ENGINEERING OF ORDNANCE. By A. T. Dawson. London, 1909. (Supplement to Vol. XIX. of the Transactions of the Junior Institution of Engineers.) (Exchange.)

FUEL ECONOMY OF DRY BLAST AS INDICATED BY CALCULATIONS FROM EMPIRICAL DATA. By R. S. Moore. (Advance paper, Iron and Steel Institute, 1909.) (Exchange.)

GEOLOGY OF THE QUEENSTOWN SUBDIVISION, WESTERN OTAGO DIVISION. (Bulletin No. 7, New Zealand Geological Survey.) By James Park. Wellington, 1909. (Exchange.)

GREAT BRITAIN. Mines and Quarries. General Report and Statistics for 1908. Pt. II.—Labor. London, 1909. (Exchange.)

"GROWTH" OF CAST-IRONS AFTER REPEATED HEATINGS. By H. F. Rugan and H. C. H. Carpenter. (Advance paper, Iron and Steel Institute, 1909.) (Exchange.)

INSTITUTION OF MINING AND METALLURGY. Bulletin No. 61. London, 1909. (Exchange.)

IRON-ORES, SALT, AND SANDSTONE. (West Virginia Geological Survey, Vol. 4.) By G. P. Grimsley. (Gift of I. C. White.)

KNOWN PRODUCTIVE OIL AND GAS FIELDS OF THE UNITED STATES IN 1908. (Map.) Washington, U. S. Government, 1909. (Exchange.)

KÖNIGLICHEN BERGAKADEMIE ZU CLAUSTHAL. Programm, 1909–1910. Leipzig, n. d. (Exchange.)

LIBRARY OF CONGRESS. Want List of Periodicals. New edition, 1909. Washington, U. S. Government, 1909. (Exchange.)

——— Want List of Publications of Educational Institutions, 1909. Washington, U. S. Government, 1909. (Exchange.)

——— Want List of Publications of Societies. New edition, 1909. Washington, U. S. Government, 1909. (Exchange.)

LO ZINCO. (Zinc: Its Properties, Ores, Metallurgy, Production, and Uses.) By R. Musu-Boy. Milano, U. Hoepli, 1909. Price, 3.50 lire. (Gift of Publisher.)

[SECRETARY'S NOTE.—Mr. Ulrico Hoepli, an enterprising publisher of Milan, has issued a series of compact modern manuals in all branches of theoretical and applied science, the catalogue of which now comprises, I believe, 1,000 titles. Of these "Hoepli Manuals," this one receives special value for non-Italian professional students from the circumstance that Italy is at present the second in the list of European countries as regards its product of zinc-ores—the first being Germany, and the product of Spain, though competing with that of Italy in tonnage, being much inferior in quality. American students, accustomed to find in French or German books their principal sources of systematic and statistical information, will be surprised by this clear, thorough, and comprehensive treatise in the Italian language. Italians also may be led to wonder that their technical literature,

which comprises many profound treatises on subjects with which their country is not industrially concerned, should have lacked heretofore a book like this. The metallurgy of zinc has recently acquired increased importance in the United States, and this contribution from Italy well deserves the attention of American practitioners.—R. W. R.]

MAGNETITE ORE-DEPOSITS AT MINEVILLE. By B. S. Stephenson. (Reprint from *Iron Trade Review*, Aug. 26–Sept. 2, 1909.) (Gift of Dr. R. W. Raymond.)

MANZANO GROUP OF THE RIO GRANDE VALLEY, NEW MEXICO. (Bulletin No. 389, U. S. Geological Survey.) By W. T. Lee and G. H. Girty. Washington, U. S. Government, 1909. (Exchange.)

MEMORANDUM RE IRON AND STEEL INDUSTRY BY THE GOVERNMENT MINING ENGINEER OF THE TRANSVAAL. Pretoria, 1909. (Exchange.)

METALLOGRAFIA APPLICATA AI PRODOTTI SIDERURGICI. (The Metallography of Iron and Steel.) By U. Savoia, Milano, U. Hoepli, 1909. (Gift of Publishers.) Price, 3.50 lire.

[SECRETARY'S NOTE.—This is another of the "Hoepli Manuals," mentioned above in my note on "Lo Zinco." But this one is less directly connected with Italian industry. At least, so we might, at first thought, declare. But on second thought, we are compelled to say that, while Italy does not rank as a leading producer of iron and steel, it is, like all civilized countries (and at a rate exceeding that of many others), coming forward as a consumer, in large quantities, of these materials; and it is the consumer, after all, who is most deeply interested in the nature of metallic products and the conditions of their manufacture. But, aside from this consideration, Italian genius has to be taken into account. In many branches of research the Italian universities and technical schools have proved that the leadership enjoyed in past centuries has not been completely surrendered, and that, when Italians are not ahead, they are at least abreast of modern progress. This little manual of siderurgic metallography is not only a proof of that proposition, but also excellent and useful, *per se*.—R. W. R.]

METASOMATIC PROCESSES IN A CASSITERITE VEIN FROM NEW ENGLAND. By L. A. Cotton. (From Proceedings of Linnean Society of New South Wales, Vol. 34, Pt. 2, 1909.) (Gift.)

MINERAL RESOURCES OF THE PHILIPPINE ISLANDS. By W. D. Smith. Manila, 1909. (Exchange.)

MINES AND METHODS. Vol. I.–date. Salt Lake City, 1909–date. (New exchange.)

MINING MAGAZINE. Vol. 1–date. London, 1909–date. (New exchange.)

NATIONAL ASSOCIATION OF CEMENT USERS. Proceedings of Fifth Annual Convention. Vol. 5. N. p., 1909. (Gift of National Association of Cement Users.)

NEW YORK CITY—BOROUGH OF BROOKLYN. Annual Report of the President, 1904, 1905, 1906, 1907. New York, 1905–1908. (Gift of President of Borough of Brooklyn.)

NONPAREIL CORKBOARD INSULATION. Montreal, 1909. (Gift of Armstrong Cork Co.)

NOTES FROM THE CHEMICAL LABORATORY. No. 2. By J. C. H. Mingaye. (Extract from Records Geological Survey New South Wales, Vol. 8, 1909.) N. p., n. d. (Exchange.)

NOTES ON EXPLOSIVE MINE GASES AND DUSTS. (Bulletin No. 383, U. S. Geological Survey.) By R. T. Chamberlin. Washington, U. S. Government, 1909. (Exchange.)

OLD MISCELLANEOUS RECORDS OF DUTCHESS COUNTY. (Second Book of the Supervisors and Assessors.) Poughkeepsie, Vassar Brothers' Institute, 1909. (Exchange.)

PRODUCTION OF COAL IN THE UNITED STATES FROM 1814, THE DATE OF THE EARLIEST RECORD, TO THE CLOSE OF 1908. (Sheet.) (Exchange.)

PRODUCTION OF IRON AND STEEL BY THE ELECTRIC SMELTING PROCESS. By E. J. Ljungberg. (Advance paper, Iron and Steel Institute, 1909.) (Exchange.)

PUBLIC BENEFITS DERIVED FROM WATER-POWER DEVELOPMENTS IN CALIFORNIA. By John Martin. N. p., 1909. (Gift of Author.)

PURIFICATION OF SOME TEXTILE AND OTHER FACTORY WASTES. (Water-Supply Paper No. 235, U. S. Geological Survey.) By H. Stabler and G. H. Pratt. Washington, U. S. Government, 1909. (Exchange.)

RELATIONS BETWEEN LOCAL MAGNETIC DISTURBANCES AND THE GENESIS OF PETROLEUM. (Bulletin No. 401, U. S. Geological Survey.) By G. F. Becker. Washington, U. S. Government, 1909. (Exchange.)

REPORT ON GRAHAM ISLAND, BRITISH COLUMBIA. By R. W. Ells. Ottawa, 1906. (Exchange.)

REPORT ON THE ESTABLISHMENT OF STATE SMELTING WORKS IN QUEENSLAND. By C. F. V. Jackson. Brisbane, 1909. (Gift of Author.)

REPORT ON THE INVESTIGATION OF AN ELECTRIC SHAFT FURNACE, DOMNARFVET, SWEDEN, etc. By E. Haanel. Ottawa, 1909. (Exchange.)

SERVICEABLE LIFE AND COST OF RENEWALS OF PERMANENT WAY OF BRITISH RAILWAYS. By R. P. Williams. (Advance paper, Iron and Steel Institute, 1909. (Exchange.)

STRUCTURAL MATERIALS IN PARTS OF OREGON AND WASHINGTON. (Bulletin No. 387, U. S. Geological Survey.) By N. H. Darton. Washington, U. S. Government, 1909. (Exchange.)

SUGGESTED RULES FOR RECOVERING COAL-MINES AFTER EXPLOSIONS AND FIRES. Ry W. E. Garforth. New York, D. Van Nostrand Co., 1909. Price, $1.50 net. (Gift of Publishers.)

[SECRETARY'S NOTE.—This suggestive little book originated in a paper read last June before the first International Life-Saving Congress at Frankfort, Germany. It contains two parts, the first dealing with suggested precautions to be taken before an accident, and the second with measures *after* an accident. There is a supplementary description (originally written in 1897 for the Institution of Mining Engineers, and here republished in full, and practically without change) of the recovery of the Altofts pit, in Yorkshire, after the explosion of Oct. 2, 1886. The story was well worth re-telling, even after the lapse of so many years, for, in its clearness and comprehensiveness as well as by reason of the circumstances of the case which it describes, it is a professional classic. Moreover, through the admirable device of marginal references to the rules suggested in previous pages, it becomes an invaluable commentary on those rules, enabling the reader, as the author pithily says, "to judge for himself of this appropriateness and utility, without waiting for another explosion to take place."—R. W. R.]

SURFACE WATER-SUPPLY OF THE UNITED STATES, 1907–08. Pt. 2—South Atlantic Coast and Eastern Gulf of Mexico. (Water-Supply Paper No. 242, U. S. Geological Survey.) Washington, U. S. Government, 1909. (Exchange.)

TESTS OF CAST-IRON. By E. Adamson. (Advance paper, Iron and Steel Institute, 1909.) (Exchange.)

TIGHT COOPERAGE STOCK, 1908. (U. S. Census Bureau. Forest Products No. 6.) Washington, U. S. Government, 1909. (Exchange.)

UNDERGROUND WATER-RESOURCES OF CONNECTICUT. (Water-Supply Paper No. 232, U. S. Geological Survey.) Washington, U. S. Government, 1909. (Exchange.)

UNIFORM MOISTURE IN BLAST. Experiment Carried Out at Clarence Ironworks, 1909. By G. Jones. (Advance paper, Iron and Steel Institute, 1909. (Exchange.)

UNIVERSITY OF OREGON, Catalogue and Announcements, Correspondence Study Department, 1909–1910. Eugene, 1909. (Gift.)

UNIVERSITY OF THE UNITED STATES, 1896, 1902. (54th Congress, 1st session. Senate. Report Nos. 429, 945.) Washington, U. S. Government, 1896, 1902. (Gift.)

——— Mr. Kyle submitted the following Reply to Views of the Minority by the Chairman of the National University Committee of One Hundred. (54th Congress, 1st session. Senate. Report 429, Pt. 3.) Washington, U. S. Government, 1896. (Gift.)

WAIHI GOLD MINING COMPANY, LTD. Report of the Directors and Statement of Accounts, 1908. London, 1909. (Gift of C. Rhodes.)

——— Report of the Proceedings at the Ordinary General Meeting of the Shareholders, May 13, 1909. London, 1909. (Gift of C. Rhodes.)

WEST VIRGINIA GEOLOGICAL SURVEY. Iron-Ores, Salt, and Sandstones. Vol. 4. Morgantown, 1909. (Exchange.)

This volume contains descriptions and analyses of all the principal iron-ore deposits of the State; a history of the old charcoal industry; descriptions and official tests or analyses of building-stones, glass sands, brines, etc. The Report is illustrated with 24 plates and 16 figures and maps. Price, post-paid by the Survey, $2 when ordered separately; or $2.25 with coal, oil, gas, and limestone map.

WESTERN AUSTRALIA MINES DEPARTMENT. Report, 1908. Perth, 1909. (Exchange.)

WHO'S WHO IN MINING AND METALLURGY, 1908. London, Mining Journal, 1908. (Gift.)

YAKUTAT BAY REGION, ALASKA. (Professional Paper No. 64, U. S. Geological Survey.) By R. S. Tarr and B. S. Butler. Washington, U. S. Government, 1909. (Exchange.)

ZEITSCHRIFT FÜR ANORGANISCHE CHEMIE. Vol. 63. Hamburg und Leipzig, 1909. (Purchase.)

GIFT OF ALUMINUM COMPANY OF AMERICA.

ALLOYS OF ALUMINUM. 1909.
FABRICATED ALUMINUM. 1909.
METHODS OF WORKING ALUMINUM. 1909.
PROPERTIES OF ALUMINUM. 1909.
USEFUL TABLES. 1909.

TECHNISCHEN HOCHSCHULE "FRIDERICIANA" ZU KARLSRUHE. (EXCHANGE.)

BAUGESCHICHTE DER STADT BRUCHSAL. Vom 13 bis 17 Jahrhundert von Roman Friedrich Heiligenthal. Heidelberg, 1909.

BAUGESCHICHTE VON KARLSRUHE, 1715–1820. Karlsruhe, 1908.

BEITRAG ZUR ELEKTROLYSE DER ALKALISALZE IM FESTEN ZUSTANDE. München, 1909.

BERICHT UBER DAS STUDIENJAHR, 1907–1908. Karlsruhe, n. d.

EINIGE GASREAKTIONEN METHANBILDUNG UND KOHLENOXYD-KOHLENSÄURE-GLEICHGEWICHT. München, 1908.

EINPHASIGE KOMPENSIERTE NEBENSCHLUSSMOTOR. Berlin, 1908.

ELEKTRIZITÄTSLEITUNG IN METALLEN UND AMALGAMEN. Temesvar, n. d.

ELEKTROMOTORISCHE VERHALTEN DES EISENS MIT BESONDERER BERÜCKSICHTIGUNG DER ALKALISCHEN LÖSUNGEN. Karlsruhe, 1909.

EXPERIMENTELLE UNTERSUCHUNG DER KOMMUTATION. Berlin, 1909.

EXPERIMENTELLE UNTERSUCHUNG EINES WECHSELSTROMSERIENMOTORS. Berlin, 1908.

FERDINAND REDTENBACHER. Bericht uber die Feier seines 100 Geburtstages. Karlsruhe, 1909.

FESTCHRIFT ZUR FEIER DES ZWEIUNDFÜNFZIGSTEN GEBURTSTAGES SEINER KÖNIGLICHEN HOHEIT DES GROSSHERZOGS FRIEDRICH II. N. p., n. d.

HYDROCYKLISCHE A-AMINOSÄUREN. Karlsruhe, 1908.

KANN EIN ELEMENT SOWOHL POSITIVE WIE NEGATIVE JONEN BILDEN? Potsdam, n. d.

METHANGLEICHGEWICHT, DIE BEZIEHUNGEN ZWISCHEN NICKEL UND WASSERSTOFF UND EINIGE METHANSYNTHESEN MIT CALCIUMHYDRÜR. München, 1909.

NEUE GALVANISCHE ELEMENTE. Berlin, 1908.

DAS NITROACETONITRIL. Karlsruhe, 1908.

NUTZBARE LEISTUNG VON GÜTERZUG-LOKOMOTIVEN UND IHR VERHÄLTNIS ZUR KOLHENDRUCK-LEISTUNG. Von E. Jacobi. Weisbaden, 1908.

OXALMALONSÄUREESTER. Greifswald, 1909.

PHYSIKALISCH-CHEMISCHE STUDIEN AN EISENSALZEN. Leipzig, 1908.

PROGRAMM FUR DAS STUDIEN JAHR, 1909–1910. Karlsruhe, 1909.

REGELUNG VON AUTOMOBILMASCHINEN. Von R. Lutz. Berlin, 1909.

REIBUNG VON DYNAMOBÜRSTEN. Berlin, 1908.

REINDARSTELLUNG BEKANNTER UND NEUER SUBHALOIDE. Berlin, 1908.

STICKOXYDBILDUNG AUS LUFT . . . Karlsruhe, 1909.

SYNTHETISCHE THEORIE DER CLIFFORDSCHEN PARALLELEN UND DER LINEAREN LINIENÖRTER DES ELLIPTISCHEN RAUMES. Leipzig, 1909.

UEBER DIE EINWIRKUNG VON KOHLENOXYD AUF NATRONLAUGE. Berlin, 1908.

UNTERSUCHUNGEN UBER DIE VARIATION DER KONSTANTEN IN DER MECHANIK. Leipzig, 1909.

VERSAGEN DER WEISSTANNENVERJÜNGUNG IM MITTLEREN MURGTALE. Ludwigsburg, 1909.

VERSUCHE ZUR DARSTELLUNG VON DIPHTALOYLCARBAZOLEN. Helsingfors, 1908.

VERSUCHE ZUR ERKENNTNIS DER MILCHSÄUREGÄRUNG. Baden, 1909.

WENDEPOLSTREUUNG UND IHRE BERECHNUNG AUF GRUND EXPERIMENTELLER UNTERSUCHUNG. Berlin, 1909.

TRADE CATALOGUES.

AUTOGRAPH REGISTER CO., Hoboken, N. J. Autographic Registers for store-rooms, warehouses, stock-rooms, cash sales, and other time- and labor-saving purposes.

CRANE CO., Chicago, Ill. Sample and descriptive bulletin of "Klingerit" packing, used by the company on all their flanges, stuffing-boxes, valves, gaskets, etc.

D. D. DEMAREST Co., San Francisco, Cal. Pacific Stem-Guides for guiding the stamp-stem in stamp-mills for crushing ores. Bulletin has complete description of the new patent and devices, making these guides more efficient than those used at the present time.

GENERAL ELECTRIC Co., Schenectady, N. Y.

White Core, 30 per cent. Para Wires and Cables, N. E. code thickness of insulation.

Tricoat Wires and Cables, N. E. code thickness of insulation.

Complete catalogue of prices, dimensions, and styles of Panel Boards and Cabinets for N. E. code standard enclosed or open link fuses.

Bulletin No. 4676, Aug., 1909. Multiple Enclosed Arc Lamps, with cuts and descriptions of different styles, and of offices, stores, and factories where they have been successfully used.

Bulletin No. 4682, Aug., 1909. Type F Form P-3 Oil Break Switches for spinning-frames and machine tools.

GOLDEN-ANDERSON VALVE SPECIALTY Co., Pittsburg, Pa. Catalogue No. 12, of cushioned, non-return valves, steam-traps, blow-offs, automatic stand-pipes, etc.

PEERBOOM & SCHURMANN, Düsseldorf, Germany. Catalogue and price-list of Tachometers and Tachographs, instruments which record independently and immediately the number of revolutions per minute of steam-engines, turbines, pumps, centrifugal appliances, and electrical machinery.

STURTEVANT MILL Co., Boston, Mass. Catalogue of plate-steel crushers for rock- and ore-crushing.

WABASH CABINET Co., 296 Broadway, New York, N. Y. Folder showing various styles of letter-files, document-files, vertical filing-cabinets, and other office filing-appliances.

United Engineering Society Library.

ENGINEERS' CLUB OF CINCINNATI. Secretary's Report and List of Members, 1890–1905, 1907–1909. Cincinnati, 1890–1905, 1907–1909.

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members or associates who accepted election during the month of October, 1909:

Members.

John Barry, ...	Animas Forks, Colo.
Thomas H. Bentley, ...	Socorro, N. M.
D. Owen Brooke, ...	Birdsboro, Pa.
Herbert L. Eaton, ...	Bob, Nev.
John W. Gankroger, ...	Indé, Durango, Mex.
Carl E. Grunsky, ...	Poto, Peru, S. America.
Ole G. Hoaas, ...	Wardner, Idaho.
Norman L. Jenks, ...	Butte, Mont.
Leslie M. Kozminsky, ...	New York, N. Y.
Aquila C. Nebeker, ...	Topliff, Utah.
John T. Roberts, Jr., ...	Buffalo, N. Y.
Joseph H. Rodgers, ...	Seattle, Wash.
Branch E. Russell, ...	Nacozari, Sonora, Mex.
Frank P. Zoch, ...	Cold Spring-on-the-Hudson, N. Y.

Associate.

Stephen Royce, ...	Cambridge, Mass.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed for election as members of the Institute during the month of October, 1909. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

Christopher Leslie Bruns, Jr., ... New York, N. Y.
Robert B. Carnahan, Jr., ... Middletown, Ohio.
Edwin E. Carpenter, ... Syracuse, N. Y.
William Wathen Charles, ... Tonopah, Nev.
John Philip Furbeck, ... Ophir, Colo.
Charles B. Hollis, ... Randolph, Vt.
Andrew Morton Howat, ... Mammoth, Cal.
James H. Polhemus, ... Carthage, Mo.
William Clark Joseph Rambo, ... Parral, Chihuahua, Mex.
Alexander Sharp, ... Orient, Wash.
Charles Arthur Stewart, ... Ithaca, N. Y.
Alfred Wood Stickney, ... Cambridge, Mass.
Malcolm Macfarlan Thompson, ... Ridgefield, N. J.
William Rogers Wade, ... Tyrone, N. M.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of October, 1909. This list, together with the lists given in the *Bulletin*, Nos. 26 to 34, for February to October, therefore, supplements the annual list of members corrected to Jan. 1, 1909, and brings it up to the date of Nov. 1, 1909. The names of Members who have accepted election during the month (new members), are printed in *italics*.

AERTSEN, GUILLIAEM..........................Railway Steel Spring Co., Latrobe, Pa.
ANDREW, THOMAS..................Rand Club, Johannesburg, Transvaal, So. Africa.
AROZARENA, R. M. DE.....................3a Estaciones No. 1, Mexico City, Mexico.
BAILEY, MELBOURNE, Min. Engr.........................Barkerville, B. C., Canada.
BARROWS, WALTER A., JR..........................1185 East Blvd., Cleveland, Ohio.
**Barry, John*, Mine Supt..Animas Forks, Colo. '09.
**Bentley, Thomas H.*, Student Post Graduate......................Socorro, N. M. '09.
BRADFORD, S. K..National, Nev.
**Brooke, D. Owen*..Birdsboro, Pa. '09.
BROOKS, RAYMOND........................5602 Monroe Ave., Hyde Pk., Chicago, Ill.
BROWN, H. LAWRENCE, Min. Engr...........P. O. Box 931, Cobalt, Ont., Canada.
BROWNE, ARTHUR B...1100 Logan St., Denver, Colo.
BRYDEN, CHARLES L.....................................436 Colfax Ave., Scranton, Pa.
BUDROW, LESTER R................Lucky Tiger Mining Co., Yzabal, Son., Mexico.
BURRALL, FREDERICK P.............................1004 Oliver Bldg., Boston, Mass.
CALDWELL, WILLIAM A., Care Maurie Badian, Medellin, Colombia, So. America.
CLARK, CARLE D...........................Care Hotel Westminster, Los Angeles, Cal.
COCKERELL, L. MAURICE..........................Apartado 143, Chihuahua, Mexico.
COLLINGS, BURTON I., London & Rhodesia Mining & Land Co.,
Salisbury, Rhodesia, So. Africa.
COLLINS, GLENVILLE A............................307 1st Ave., South, Seattle, Wash.
DAVIS, ARTHUR E.........................Care Dr. H. V. Jackson, Durango, Mexico.
DEEGAN, JOHN...Hemet, Cal.

DEVEREUX, WILLIAM G..........................Melones Mining Co., Melones, Cal.
DOUGLASS, ROSS E..East Ely, Nev.
**Eaton, Hubert L.*, Met......... Care Lodi Mines Co., Bob, via Luning, Nev. '09.
EDELEN, ALEXANDER W., Angangueo Unit Am. Smltg. & Refg. Co.,
Angangueo, Mich., Mexico.
EDMONDSON, HORACE W., Cia Minera de Rio Plata, S. A.,
Guazapares, Chih., Mexico.
EILERS, KARL..................................320 Central Park, W., New York, N. Y.
EMMONS, WILLIAM H., Univ. of Chicago, Walker Museum.............Chicago, Ill.
ERDLETS, JOSEPH F., JR., Care F. F. Sharpless, 52 Broadway, New York, N. Y.
FAIRBAIRN, GEORGE.................The Avino Mines, Ltd., Gabriel, Dur., Mexico.
FELL, E. NELSON..."Creedmore," Warrenton, Va.
FITZSIMMONS, F. J...............The Aguacate Mines, San Mateo, Costa Rica, C. A.
FRANK, ALBERT..Instructed to hold all mail.
**Gankroger, John W.*, Mine Supt.....Guadalupe Mine, Indé, Dur., Mexico. '09.
GARLICHS, HERMAN..............................404 W. 115th St., New York, N. Y.
GEORGE, HAROLD C., Dir., Wisconsin State Mining Trade School,
P. O. Box 166, Platteville, Wis.
GODFREY, ELI S., Supt...............................Onteora Club, Tannersville, N. Y.
GRAHAM, STANLEY N.........................506 Princess St., Kingston, Ont., Canada.
GRAVES, MACDOWELL...................Keating Gold Mining Co., Radersburg, Mont.
**Grunsky, Carl E.*, Min. Engr., Poto, via Arequipa & Cojata,
Peru, So. America. '09.
GUNTHER, C. GODFREY............................Care Santa Rita Hotel, Tucson, Ariz.
HAAS, J. CLEVELAND, Min. Engr................518 Peyton Block, Spokane, Wash.
HARGRAVES, ERNEST P., Met......Gwalia Consolidated, Wiluna, West Australia.
HERZ, NATHANIEL.......................................115 Linden St., New Haven, Conn.
HEWITT, GEORGE H...707 State St., Springfield, Mass.
HIBBS, JOSEPH G....................................1302 Penna. Bldg., Philadelphia, Pa.
HIGGINS, EDWIN, Cons. Min. Engr..................Central Bldg., Los Angeles, Cal.
**Hoaas, Ole G.*, Min. Engr.......................P. O. Box 217, Wardner, Idaho. '09.
HORE, REGINALD E., Michigan College of Mines, Dept. of Geol.,
Houghton, Mich.
HUSTON, HARRY L., Mgr.................................Keltz Mining Co., Sonora, Cal.
JACKSON, BYRON N..............................836 Kensington Rd., Los Angeles, Cal.
***Jenks, Norman L.*, Min. Engr., British Butte Mining Co., Box 367,
Butte, Mont. '09.
JONES, CHARLES C............................102 S. Occidental Bldg., Los Angeles, Cal.
JONES, HENRY E...........The Lodge, Culmington Rd., Ealing, London, England.
JONES, WM. STRICKLER...Greensburg, Pa.
KELLEY, ARTHUR L., So. American Development Co.,
Guayaquil, Ecuador, So. America.
KENDALL, HUGH F., Supt. of Mines........N. Y. State Steel Co., Virginia, Minn.
KIMBER, ALFRED........Industrial Securities Co., 25 Broad St., New York, N. Y.
**Kozminsky, Leslie M.*, Min. Engr..............2 Rector St., New York, N. Y. '09.
KUEHN, AMOR F..........................Room 1417, 71 Broadway, New York, N. Y.
KURIE, FRANK M......................................529 Drexel Bldg., Philadelphia, Pa.
LANGFORD, FRANK.... ....Care Braden Copper Co., Graneros, Chile, So. Amer.
LEAVELL, JOHN H...24 South St., Quincy, Mass.
MCBRIDE, WILBERT G., Genl. Supt., Great Western Copper Co., Courtland, Ariz.
MCCORD, WILLIAM H., Providencia Mining & Milling Co., Apartado 67,
Guanajuato, Mexico.

MCMAHAN, CHARLES H., 3a Calle del la Merced No. 1, Aguascalientes, Mexico.
MACDONALD, JESSE J..Instructed to hold all mail.
MANN, WILLIAM S., Min. and Met. Engr., Genl. Mgr., El Conde Mining Co., S. A., Tepehuanes, Dur., Mexico.
MARR, ROBERT A..Holmes, Wyo.
MATTHIESSEN, F. W............................St. Anthony Hotel, San Antonio, Texas.
MAYNARD, REA E.....................419 So. Commonwealth Ave., Los Angeles, Cal.
MEGRAW, HERBERT A., Hacienda Santa Eleua, San Luis de la Paz, Gto., Mexico.
MOORE, CHARLES J., Cons. Min. Engr..........741 Equitable Bldg., Denver, Colo.
*_Nebeker, Aquila C._, Surveyor, Assayer and Engr., Care Scranton Mines, Topliff, Utah. '09.
NOVOA, BARTOLOMÉ, Administrador General, Sociedad Minera Alpamina, Yauli, Peru, So. Amer.
O'BYRNE, JOSEPH F., Min. Engr.................. ....Contact, Nev.
ODDIE, TASKER L..Hawthorne, Nev.
PAYMAL, GEORGE W...............................1621 Vallejo St., San Francisco, Cal.
POILLON, HOWARD A.................................125 E. 70th St., New York, N. Y.
PRITCHARD, DE V. G., 100 Cape Road, Port Elizabeth, Cape Colony, So. Africa.
RAGAZ, IVAN...Esmeralda, Sierra Mojada, Mexico.
REECE, FREDERICK B...................................Socorro Mines, Mogollon, N. M.
REEVES, THOMAS V..2618 Haste St., Berkeley, Cal.
*_Roberts, John T., Jr._, Min. Engr.......... ......350 Main St., Buffalo, N. Y. '09.
*_Rodgers, Joseph H._, Min. Geol.................1601 13th Ave., Seattle, Wash. '09.
ROPP, ALFRED VON DER.......Room 307-8, 454 California St., San Francisco, Cal.
†_Royce, Stephen_, Mine Surveyor...............103 Irving St., Cambridge, Mass. '09.
RUMBOLD, WILLIAM R................Union Club, Trafalgar Sq., London, England.
*_Russell, Branch E._, Min. Engr.........Apartado 22, Nacozari, Son., Mexico. '09.
RUSSELL, WILLIAM C...Manhattan, Nev.
SAXMAN, C. W., JR., Cons. Min. Eng....312 Dooly Bldg., Salt Lake City, Utah.
SMITH, E. PERCY..29 Broadway, New York, N. Y.
SNOW, FREDERICK W.............................Chino Copper Co., Santa Rita, N. M.
STENGER, EDWARD L................................P. O. Box 423, Bellingham, Wash.
STOEK, HARRY H., Prof., Min. Engr..................Univ. of Illinois, Urbana, Ill.
SWAIN, HARRY L............Avenida Cinco de Mayo No. 1, Mexico City, Mexico.
TAFT, HARRY H..1470 Steele St., Denver, Colo.
TENNY, EMIL B.......................................6641 Vermont Ave., St. Louis, Mo.
THOMPSON, WILLIAM, Wks. Mgr. and Engr., Canada Producer & Gas Engine Co., Barrie, Ont., Canada.
THOMSON, HENRY N..........Care International Smltg. & Refg. Co., Tooele, Utah.
THURSTON, E. COPPEE....................................Booth Ave., Englewood, N. J.
VAN LAW, CARLOS W.....Cia Real del Monte y Pachuca, Pachuca, Hgo., Mexico.
WARDEN, BRUCE R., Civ. and Min. Engr., Dawson, Anderson & Warden, 536 Hastings St., Vancouver, B. C., Can.
WILLIAMS, DAVID...... ....437 Eleventh Ave., New York, N. Y.
WORTH, JOHN G....................................... ....P. O. Box 352, Reno, Nev.
*_Zock, Frank P._, Tunnel Work............Cold Spring On-the-Hudson, N. Y. '09.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address on Records, from which Mail has been Returned.
Adams, Randolph, ...	Copperhill, Tenn.
Alexander, George E., ...	Sparta, Ore.

Allen, Frederick E., ... Bloomsburg, Pa.
Andersen, Carl, ... Johnnie, Nev.
Bartoccini, Astolfo, ... 214 E. 90th St., New York, N. Y.
Bassett, Thomas B., ... Cumpas, Sonora, Mexico.
Batchelder, Joseph F., ... 54 1st St., Portland, Ore.
Bellam, Henry L., ... Reno, Nev.
Bouchelle, James F., ... 22 Duncan Ave., Jersey City, N. J.
Brook, Henry E. C., ... Cadia, N. S. W., Australia.
Brown, Frank H., ... Coppermount, Alaska.
Brown, Harvey S., ... Yerington, Nev.
Campa, Jose, ... Mexico City, Mexico.
Carver, Leonard H., ... 452 E. 16th St., Oakland, Cal.
Cragoe, A. Spencer, ... Vencedora, Mexico.
Derby, Harry S., ... 134 Monroe St., Chicago, Ill.
Dickson, George H., ... Lethbridge, Alberta, Canada.
Dougherty, Clarence E., ... 41 Wall St., New York, N. Y.
Ekberg, Benjamin P., ... Johannesburg, Transvaal, So. Africa.
Field, Wilfrid B., ... Mexico City, Mexico.
Francis, George G., ... 177 St. George's Sq., London, W., England.
Fuller, Frederick D., ... Sumpter, Ore.
Gage, Edward C., ... San Dimas, Dur., Mexico.
Gee, Emerson, ... Reno, Nev.
Hawkins, Tancred, ... Ballydehob, Ireland.
Hunt, Thatcher R., ... Iron Mt., via Keswick, Cal.
Jessop, Herbert J., ... Guanacevi, Mexico.
Jewett, Eliot C., ... 2918 Morgan St., St. Louis, Mo.
Judd, Henry A., ... Mertondale, W. Australia.
King, Rufus H., ... Union Club, New York, N. Y.
Kow, Tong Sing, ... Shanghai, China.
Mildon, Reginald B., ... Nacozari, Son., Mexico.
Moulton, Herbert G., ... Cobalt, Ont., Can.
Muir, Thomas K., ... Portland, Ore.
Nawatny, William F., ... Harrisburg, Ill.
Nelson, Edward E., ... 307 Dooly Blk., Salt Lake City, Utah.
Nissen, Peter N., ... Union Trust Bldg., Los Angeles, Cal.
Philbrick, Arthur, ... Manhattan, Nev.
Piper, John W. H., ... Buenos Ayres, Argentine Rep., S. A.
Potter, J. A., ... 41 W. 124th St., New York, N. Y.
Rigney, Thomas P., ... Reno, Nev.
Rodda, Richard W., ... Seattle, Wash.
Sandifer, Harmer C., ... El Oro, Mexico.
Schlemm, William H., ... Durango, Mexico.
Scott, Winfield G., ... Long Beach, Cal.
Skelding, Joseph F., ... Embreeville, Tenn.
Thomas, Richard A., ... 43 Wall St., New York, N. Y.
Vaux, Charles A., ... P. O. Box 80, East Rand, So. Africa.
Vidler, Louis W., ... Lookout Mountain, Colo.
Warren, Henry L. J., ... Salt Lake City, Utah.
Wiswell, Herbert J., ... Carterville, Mo.
Wolfe, Burton L., ... Ely, Nev.
Young, William, ... Kenora, Ont., Canada.

NECROLOGY.

The deaths of the following members have been reported to the Secretary's office during the month of October, 1909:

Date of Election.	Name.	Date of Decease.
1896.	*Rasmus Hanson,	September 28, 1909.
1899.	*Algernon K. Johnston,	October 3, 1909.
1903.	*Bertel Peterson,	February 10, 1909.

BIOGRAPHICAL NOTICES.

John Marriott Grice was born Nov. 7, 1880, at St. Louis, Mo. He studied three years at the Mass. Inst. of Technology and one year at the Michigan College of Mines. From December, 1903, to April, 1904, he was employed in mining and metallurgical work at the mines of Santa Maria de la Paz, Matehuala, San Luis Potosi; then for about two years in a similar capacity by the El Oro Mining & Railway Co., Ltd., and in 1906 and 1907 he superintended successively the cyanide-plants of the Mezquital gold-mines, Zacatecas, and La Atrevida, Jalisco, besides making professional examinations in Durango and in San Luis Potosi, where he became in 1907 manager of the Benito Juarez mines at Salinas. Early in 1908 he returned to El Oro, but went at the beginning of 1909 to Guanajuato, where he took a contract at the Peregrina mine. On Sept. 15, 1909, by the accidental and unexplained explosion of two and a half cases of dynamite, Mr. Grice and four native laborers were instantly killed on the fourth level of the mine. Mr. Grice was well known and highly esteemed in Mexico. He joined the Institute in 1908.

Rasmus Hanson was born in Denmark about 1847, but came to this country while a young man and settled in Colorado. After working for several years in the mines of Gilpin county, he removed in 1870 to the San Juan region, in which he was one of the earliest and most adventurous pioneers. He became a member of the Institute in 1896, and died Sept. 28, 1909, at Silverton, Colo. Mr. Hanson acquired and developed several noted and productive mines in the San Juan region, such as the Golden Fleece, Hidden Treasure, Enterprise, Mastodon and Scotia, and especially the Sunnyside Extension, from which he

* Member.

is said to have realized large profits. His friends and acquaintances describe him as a man of rugged exterior but fine character and great generosity.

Harold Heathcote Harvey was born in Montreal, Can., in 1877. Twenty-five years ago his father, LeRoy G. Harvey, moved to Oakland, Cal., where the family has been ever since. Mr. Harvey was graduated from the University of California, with the class of 1900, as civil engineer. Prior to his graduation, he had been mill-superintendent at the Soulsby mine in California. After leaving school, he spent four years in various positions as mill-superintendent or mining engineer at Sonora, Cal., Elk City, Idaho, and Tonopah, Nev. In 1903 he was connected with the engineering department of the Southern Pacific railway, and was secretary of the Tonopah Gold Belt Mining Co. In 1904 he was employed as mining and civil engineer for the Alaska Trading & Mining Co., at Teller, Alaska. During that and the following year he constructed what was at that time one of the longest ditches for mining purposes on Seward Peninsula, approximately 20 miles in length. He was later deputy mineral surveyor in the Port Clarence district.

During the winter of 1906–07 Mr. Harvey was stationed at Tin City, Alaska, and while there had a severe attack of typhoid fever, from which he never fully recovered, and which resulted in bringing to a close the career of a bright and promising engineer. The nearest doctor was at Nome, 135 miles distant. Mr. Harvey's wife was with him, and it was due to her careful nursing that he pulled through and was able to return again to California. To be in a bleak and desolate country, locked in on all sides with snow and ice, and nothing but the dazzling whiteness of the glaring snow for the eyes to gaze upon for eight months in the year, is indeed a hardship for a sick man. Mail arrived once a week, and this was the only means of getting medicine, after consulting a doctor by telephone. The winter at Tin City was severe, and during those long, dreary, cold months on Cape Prince of Wales, between the Bering sea and the Arctic ocean, Mr. Harvey convalesced so far he could be up all the time, and, although yet weak, he set out, in company with Mrs. Harvey, by dog-sled to Teller, Alaska, a distance of

60 miles, where the remaining three months of winter were spent. They returned to California the following summer.

A short abstract from a recent letter of Mrs. Harvey's is not out of place in this notice:

"A truly happy married life is something to be thankful for, and that mine was, for I have not one single unhappy thing to look back upon. Harold was absolutely everything to me, husband, lover, friend, and child, for at the last he was in bed five months, and I took care of him night and day. Business prospects were quite bright for him last winter, and he seemed in very fair health, but took a severe cold in the spring, while looking up a new 'strike' in Nevada, which went into grippe and then into rapid tuberculosis. He suffered intensely at times, but was ever brave, cheery, and unselfish, using his brains to the last."

He was not a strong man physically, but had a large amount of vitality, was energetic, and would not give up. Even when he had gone to bed for the last time, he worked for several days finishing up a report on a copper-mine that he had recently examined.

Although only 32 years old, he had a wide field of experience, and was rapidly making his way to the front. He was a member of the Delta Upsilon Fraternity, the Live Oak Lodge, No. 61, F. and A. M., and became a member of the American Institute of Mining Engineers in 1903. His widow, formerly Miss Edith Goodfellow, of Oakland, survives him. He died Aug. 24, 1909, at Oakland, California. [The foregoing notice was furnished by Mr. Albert H. Fay, New York, N. Y.]

Bertel Paterson was born Dec. 3, 1868, at Paterson, N. J., and educated in the public schools and high schools of that town. Upon his graduation in 1886, he went with his uncle, Mr. James Baxter, a mining engineer, to Villaldama, Nuevo Leon, Mexico, where he was for four years connected with the Guadalupe Mining Co. as an assayer, and afterwards as mining superintendent. In 1890 and 1891, he was connected with the Villaldama Mining Co., in the same State, as assayer and surveyor in charge of underground work. From 1892 to 1898, he was employed at different times as mining superintendent of various mines in Coahuila and Nuevo Leon, operated by M. Guggenheim's Sons. In 1899, he was made General Manager of the Grand Central Mining Co. at Torres, Sonora, Mexico. But two or three years afterwards he decided to give his attention to the development of a number of mining-properties in

which he was interested in the States of Queretaro and Chihuahua. He was actively engaged in the management of one of these properties at El Rayo, near Santa Barbara, Chihuahua, when he contracted a fever which proved fatal. He died at the hospital in Los Angeles, Cal., Feb. 10, 1909. His body was brought to his native town, Paterson, N. J., for burial. Mr. Paterson was known throughout Mexico and the United States as "Bert" Paterson, and was highly esteemed professionally and socially. He was a lifelong and consistent member of the Fourth Baptist church in Paterson, and both there and elsewhere was recognized as a man of upright and generous personal character. He became a member of the Institute in 1900.

Robert Pitcairn was born May 6, 1836, at Johnstone, Scotland, and came as a child to this country. In 1848, after a limited common school education, he entered the service of the Atlantic & Ohio Telegraph Co. as a messenger, and was soon promoted to be an operator. In 1853, he became telegraph operator of the Pennsylvania railroad and assistant ticket agent at Duncansville, Pa. The next year he was transferred to the office of the General Superintendent, where he served successively as clerk, Superintendent of the Telegraph, and Superintendent of the Pittsburg Division. The latter position he occupied from 1866 to 1902, being thus for thirty-six years the active head of the Pennsylvania railroad and its branches between Pittsburg and Altoona. From 1876 to 1902, he was also General Agent of the Pennsylvania Railroad Co., and closely identified with the introduction and development of many marvelous improvements for the protection of employees and the comfort and safety of the traveling public.

In 1902, he was appointed by the Board of Directors, Resident Assistant to the President at Pittsburg, a position which he held with credit until May 6, 1906, the seventieth anniversary of his birth, when he was retired for age, according to the invariable rule of the company.

Mr. Pitcairn was Vice-President of the Westinghouse Air Brake Co., and held other positions of trust and responsibility.

He died July 25, 1909, leaving behind him an enviable record of well-deserved success and of universal esteem. He became a member of the Institute in 1881.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Ventilating-System at the Comstock Mines, Nevada.

BY GEORGE J. YOUNG,* RENO, NEV.

(Spokane Meeting, September, 1909.)

CONTENTS.

I. INTRODUCTION.

DR. JOHN A. CHURCH, in his treatise on the Comstock Lode,[1] gave a full and clear account of the conditions of the mine

* Professor of Mining and Metallurgy, University of Nevada.

[1] *The Comstock Lode: Its Formation and History* (1879).

during the period of greatest activity. The difficulties in the way of deep mining at that time were excessive water and high temperature. The drainage of the mines taxed the financial resources of the mining companies, and the high temperatures restricted the capacity of the miners for underground labor. In spite of these difficulties exploratory work was continued until the financial burden of removing the water became too great. The companies then restricted the work to the upper levels. The years 1883 to 1886 marked the period of cessation of deep mining.

For 12 years thereafter mining was confined to the levels above the Sutro tunnel; and, while the question of drainage was more or less agitated, nothing of importance was done until the formation of the Comstock Pumping Association in 1898. Since that time successive levels have been drained until, at present, on the north-end mines, a depth of 700 ft. below the Sutro-tunnel level has been reached. Mining is now carried on in the Ophir mine from the 1,700- to the 2,200-ft. levels, and a winze is being sunk to make connection with the 2,300-ft. level (the 2,450 ft. of the C. & C. shaft).

In 1903 the Ward Shaft Association was formed to undertake the draining of the Central and Gold Hill group of mines, and at the present time a depth of 2,500 ft. has been reached in the Ward shaft. The situation at the Ward looks promising, and the 3,000-ft. level may be reached in the near future. It is proposed to establish a large pumping-station on this level, and then begin the opening of known ore-bodies and the prospecting for new ones.

The main factors which have contributed to the successful solution of the drainage problem are the Sutro tunnel, the use of the hydraulic elevator as a sinking-pump, the concentration of pumping-units, and cheap electrical power.

The question of high temperature and its control still remains as a minor but important problem. The temperatures encountered in the new workings are as great as those in the earlier periods. The Comstock was considered the hottest group of mines in the world, and I am not aware of similar conditions in any other mining-district. A water-temperature as high as 160° F. has been reported from the Ward shaft, and temperatures below the Sutro tunnel are very high in many instances.

The control of excessive underground temperatures is effected by proper ventilation, or by the provision and distribution of air-currents of sufficient volume. Upon the Comstock of late years considerable attention has been paid to ventilation, and underground conditions have been much improved.

The ventilation of metalliferous mines has received but little attention, and comparatively few data are available. The literature of coal-mining, on the other hand, is replete with data of mine-ventilation, the topic being one of considerable interest even at the present time. In coal-mines, ventilation has for its principal object the removal of explosive gases. To accomplish this, fans are generally employed to force into or exhaust from a mine large volumes of air. In metal-mining, while occurrences of gases are not infrequent, the volume of gas encountered is usually so small as to be unimportant, and it is generally non-explosive. In these cases the purpose of the ventilating-current is to supply pure air to the miners in the different working-places, and to remove the gaseous products of blasting. In but few instances has the ventilating-current to perform the additional function of cooling the mine-workings. In most metal-mines ventilating-currents are established by the use of two shafts, one an upcast, the other a down-cast. The natural elevation of the ground-temperature, due to increased depth, is relied upon to warm the air sufficiently to start an upward current. As depth is attained, the average temperature of the ascending current is raised and the ventilating-efficiency of an upcast shaft increased. Fans and air-pipes are used for the ventilation of pits, shafts, and dead-ends, but are seldom used for the general ventilation of a large mine. The Comstock mines furnish an excellent example of the use of natural ventilation for providing a current of sufficient volume partly to overcome abnormal conditions. That natural ventilation is insufficient and not controllable enough, is evidenced by the fact that at the present time one large fan is in operation and another is being erected. Ventilation by small fans is also very largely used to supplement the effect of the shafts. The purpose of this paper is to give a detailed account of the system of ventilation at present in use at the Comstock mines.

II. Temperatures.

In any system of natural ventilation, surface and underground temperatures play an important part.

1. *Surface-Temperatures and Conditions.*—Virginia City, Nev., does not possess a meteorological station, and consequently no records of the weather-conditions are available. However, the records of the government station at Reno, 18 miles NW., supply this deficiency. Virginia City has an altitude of between 6,000 and 6,500 ft., and Reno 4,553 ft., above sea-level.

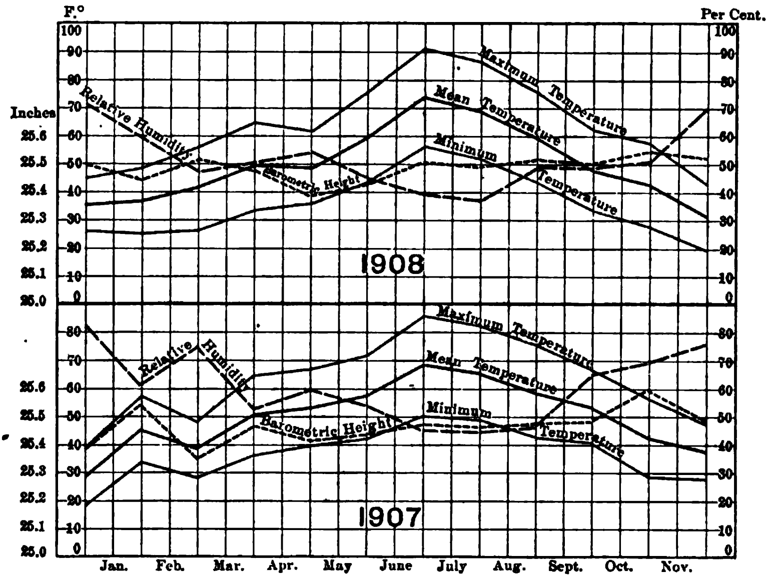

Fig. 1.—Curves of Maximum, Mean, and Minimum Temperatures, Relative Humidity, and Barometic Height, at Reno, Nev., in 1907 and 1908.

The difference in altitude would make some, but not a very great, difference in climatic conditions.

The maximum, mean, and minimum temperatures, the barometric height, and the average relative humidities for the years 1907 and 1908, are plotted in Fig. 1. It should be noted that the day-humidity is much less than the average 24-hr. humidity; for instance, in July, 1907, the average humidity of four days was 23; in August, the average of four days was 19; in September, the average of two days was 21.5; in October, the average of four days was 51; in November, the average of three days was 43; and in December, the average of four days

was 64 per cent. The average day-humidity during the months of July, August, and September was 21; and for the months of October, November, and December it was 51 per cent. The air during the summer and fall months is characterized by a low day-humidity, and during the winter months the day-humidity is about normal. The average yearly temperature is about 50° F.

2. *Underground Temperatures.*—Becker[2] has given excellent data on the rock-temperatures prevailing at different points underground, and to this authority the reader is referred. Air-temperature only will be considered in this paper.

Records of air-temperature in the Ophir mine have been kept for some time, and from these records have been compiled Tables I., II., III., and IV.

TABLE I.—*Temperatures at Sutro-Tunnel Level.*

Date, 1908.	No. 1, South of Cross-cut to C & C. Shaft.	No. 2, South of Winze.	No. 3, Head of Winze.	No. 4, From Union Connection to Ophir Incline.
	Degrees F.	Degrees F.	Degrees F.	Degrees F.
June 4	85	102	104	100
June 5	90	102	102	100
June 6	87	100	104	98
June 8	90	100	100	90
June 10	92	100	104	.....
June 11	94	100	.....	.....
June 15	94	98	.....	98
June 24	96	.....	.....	.....
July 2	96	102	102	96
July 8	96	100	.....	.....
July 16	97	102	104	97
July 17	98	102	105	96

Table I. shows the temperatures measured in the north lateral of the Sutro tunnel during June and July, 1908. The hottest months are July and August, during which the conditions in the main Sutro tunnel in the vicinity of the Combination shaft are particularly trying. No. 1 station in the north lateral (about 500 ft. south of the connection to the C. & C. shaft) shows an average temperature of 91° F. for June and 96.7° for July. Only a feeble air-current was moving towards the Sutro tunnel at this station. At No. 2 station, between

[2] Geology of the Comstock Lode and the Washoe District, *Monograph III.*, *U. S. Geological Survey*, pp. 228 to 265 (1882).

the C. & C. shaft connection and the "hot winze," the air was practically dead, and in June the average was 100.3°; for July it was 101.6°. On July 17, I measured a temperature of 102° at this point. No. 3 station, at the head of the "hot winze," delivering hot air from the workings of the Ophir mine below this level, gave an average June temperature of 102.8°, and for July, 103.3°. No. 4 station, at the connection with the Union shaft, showed an average temperature of 95.2° for June and 96.3° for July. The lowering of the temperature at No. 4 station was due to the cool air from the Union shaft leaking through a canvas curtain and joining the hot air at both the north lateral and the connection to the Union shaft. At stations Nos. 3 and 4 the temperature was measured in a swiftly-moving air-current.

In the Sutro tunnel, south lateral, no temperature-records were available, but observations made by me showed a temperature of 108.9° in the vicinity of the Julia shaft, Dec. 13, 1908; in the Ward shaft connection, 101.1°; at the junction of the Ward shaft connection and the south lateral, 86°; at the connection of the Sutro tunnel and south lateral, 100°; west of the Combination partition, 94.6°, and east of the partition, 93.7°. A temperature has been noted[3] of 105° for Apr. 15, 1905, in the Sutro tunnel, east of the Combination partition, and during the months of July and August, 1908, this temperature is reported to have prevailed.

The Sutro tunnel is an incast and, as a consequence, the temperatures vary from a minimum at the portal to a maximum at the Combination partition. This variation was measured, and the results are shown in Table II. The tabulated results are plotted in Fig. 2.

An air-temperature of 46° was raised to 95° in passing a distance of 18,000 ft. The air-temperature in the Sutro tunnel varies with the temperature of the outside air, the temperature of the water passing through, and the rock-temperature. The most important factor is the temperature of the water pumped from the C. & C. and the Ward shafts. This water is passed through both the north and south laterals in 24-in. wooden-stave pipes, and at a point 1,000 ft. east of the Combination partition (approximately 18,000 ft. from the portal) the pipe terminates

[3] Leon M. Hall and Frederic W. Bishop, *Report to the Comstock Pumping Association* (May 25, 1908).

TABLE II.—*Variations of Temperature in Sutro Tunnel.*

Date.	Distance from Portal.	Temperature of Water.	Temperature of Air.	Relative Humidity.	Notes.
	Feet.	Deg. F.	Deg. F.	Per Cent.	
Nov. 22, 1908.	Combination partition.	......	94.5	98.5	
Nov. 22, 1908.	18,000	......	94.4	98.2	Near Combination partition.
Nov. 22, 1908.	17,000	95.0	95.0	97.2	End of wood pipe.
Nov. 22, 1908.	13,000	......	95.0	93.7	
Nov. 22, 1908.	10,000	93.7	91.9	98.0	Prospect switch.
Nov. 22, 1908.	8,000	92.8	85.2	92	
Nov. 22, 1908.	6,800	90.7	73.0	83	Beginning of 30-in. pipe.
Nov. 22, 1908.	5,200	......	71.6	74	
Nov. 22, 1908.	0	......	46.0	59	
Dec. 13, 1908.	E. of Combination partition.	......	93.74	96.4	
Dec. 13, 1908.	16,400	92.5	93.2	100	
Dec. 13, 1908.	15,000	94.1	93.56	97.2	

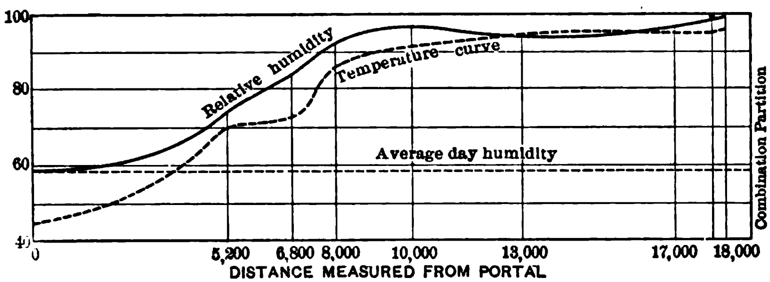

FIG. 2.—CURVES SHOWING TEMPERATURE AND RELATIVE HUMIDITY VARIATION IN THE SUTRO TUNNEL.

and the water flows out, occupying about two-thirds and in some cases the full width of the tunnel. From the portal a 30-in. stave pipe has been constructed a distance of 7,000 ft., and between this point and the former, or a distance of 11,000 ft., the air flows over the surface of the water. This exposure not only heats the air but also saturates it with water-vapor. The close coincidence between the air-temperature and the water-temperature should be noted.

The high temperature in the south lateral near the Julia shaft is due to rock-temperature more than to water, since in the south lateral the water is carried in a wooden pipe. The temperature-measurements in the north and south laterals are shown in Table III.

TABLE III.—*Temperature-Measurements in North and South Laterals of Sutro Tunnel.*

Date.	Distance.	Temperature of Water.	Temperature of Air.	Relative Humidity.	Notes.
		Deg. F.	Deg. F.	Per Cent.	
Dec. 13, 1908.	Surface, Ward shaft......	......	40.1	39.6	7.45 a.m.
Dec. 13, 1908.	1,600-level station.........	......	88.7	65.5	
Dec. 13, 1908.	Half to south lateral......	......	101.1	63.5	
Dec. 13, 1908.	Junction south lateral and Ward connection.	......	87.8	80.4	
Dec. 13, 1908.	South lateral south from charge station.	......	86	84.6	
Dec. 13, 1908.	2,300 ft. from tunnel......	......	96.8	64.4	
Dec. 13, 1908.	1,550 ft. from tunnel......	......	105.8	54.4	
Dec. 13, 1908.	400 ft. from tunnel.........	......	108.86	55.9	
Dec. 13, 1908.	100 ft. from tunnel.........	......	104.0	63.8	
Dec. 13, 1908.	Sutro tunnel at junction of south lateral.	.....	100	73.7	
Dec. 13, 1908.	Sutro tunnel 100 ft. west of north lateral.	......	87.8	91.4	
Dec. 13, 1908.	75 ft. west of Combination partition..........	......	94.6	83.9	
Oct. 11, 1908.	Outlet of air-pipe near winze, north lateral.	......	97.0	46.5	
Oct. 11, 1908.	Cross-cut to C. & C. shaft....................	95.9	93.2	95.0	
Oct. 11, 1908.	Just before reaching Mint shaft..............	......	91.4	95.0	
Oct. 11, 1908.	Junction of Sutro tunnel and N. lateral....	......	90.9	94.0	

3. *Water-Temperatures in Sutro Tunnel.*—Opportunities for measuring the water-temperature are few and far between. On Oct. 11, 1908, the water discharged from the C. & C. shaft into the tunnel gave a temperature of 95.9° F.; on Dec. 12 the water from the Ward shaft, close to the Ward, gave a temperature of 141.8°. On the same day, at a point 15,000 ft. distant from the Sutro-tunnel portal, the water-temperature measured 94.1°. The water-flow from the C. & C. shaft amounts to from 4,000 to 4,500 gal., and from the Ward, from 600 to 800 gal. per min. In addition, all the water from the ground above the Sutro tunnel drains into the laterals and discharges into the Sutro tunnel. The average temperature of the water flowing into the Sutro tunnel at the Combination partition is between 94° and 95°. Miners say that sometimes this temperature is exceeded.

4. *Temperatures of Mine-Workings.*—The temperatures in the Ophir workings have been observed for some time, and from

TABLE IV.—*Temperature-Measurements in Different Parts of the Ophir Mine.*

Date, 1907.	2,250 Cross-cut Station.	2,200 Snow-sheds.	2,200 Fan.	2,100 Ophir.	2,100 N. Stopes	2,200 Station.	2,200 Raise.
	Deg. F.	Deg. F.	Deg. F.	Deg. F.	Deg. F.	Deg. F.	Deg. F.
July 13	75	104	98	98	98	100	111
July 16	68	104	97	98	97	98	110
July 22	78	104	97	97	98	100	111
July 29	72	105	98	98	98	100	110
Aug. 7	72	110	98	98	98	99	111
Aug. 12	68	106	98	98	97	102	111
Aug. 20	66	108	97	98	97	105	113
Aug. 30	69	109	98	98	96	105	112
Sept. 12	66	104	97	98	97	101	112
Sept. 26	68	102	97	96	96	100	111
Oct. 3	62	104	96	97	98	100	110
Oct. 14	70	105	99	98	100	102	112
Oct. 24	64	103	98	101	102	102	110
Oct. 30	62	101	97	100	102	102	109
Nov. 9	58	97	94	97	100	101	110
Nov. 20	54	96	92	96	100	98	108
Nov. 25	55	100	87	95	100	98	108
Dec. 2	57	102	91	97	100	98	108
Dec. 9	57	109	92	99	100	99	109
Dec. 21	56	107	91	96	100	101	109
Dec. 26	57	109	92	96	100	101	108
Jan. 1, 1908	55	106	90	95	100	100	108
Jan. 15, 1908	60	107	91	95	100	100	108
Feb. 1, 1908	54	107	89	94	100	100	108
Feb. 18, 1908	56	112	93	91	101	102	108
Mar. 5, 1908	56	105	92	94	100	102	108
Mar. 31, 1908	57	108	93	96	100	104	108

TABLE V.—*Temperature-Readings in Different Parts of Ophir Mine and on Different Dates.*

Date, 1908.	2,250 Cross-cut Station.	2,250 Drift.	2,350 Drift.	2,100 N. Drift.	2,100 Stopes.	2,200 N. E. Drift.	2,200 Stopes.	2,200 Winze.
	Deg. F.	Deg. F.	Deg. F.	Deg. F.	Deg. F.	Deg. F.	Deg. F.	Deg. F
Apr. 27	63	106	130	104	103	104	105	106
May 20	64	112	112	101	102	101	104	103
May 23	64	114	129	101	102	101	104	102
May 28	65	112	128	100	101	100	102	102
June 1	65	112	126	100	101	101	103	101
June 5	62	110	127	100	101	99	101	101
June 6	62	108	126	100	101	99	101	100
July 20	74	100	102	101	104	104	105	102
July 23	75	100	101	101	102	100	102	100
July 25	75	104	104	102	103	101	102	100
July 30	77	105	110	102	102	105	103	100
July 31	76	101	110	102	105	104	104	101
Aug. 1	78	102	112	102	103	104	104	102
Aug. 3	76	100	110	101	103	104	104	102
Aug. 4	75	106	112	101	104	104	104	101
Aug. 5	74	100	112	101	102	102	102	101
Aug. 3, 6, 7	72	100	112	103	102	102	102	102

the daily reports Tables IV. and V. have been compiled. Only a few readings were tabulated, since but very little variation in the temperature takes place from day to day.

From Tables IV. and V. the following averages have been calculated for workings below the Sutro tunnel:

Working-Place.	Average from all Tables.	Average Maximum from all Tables.	Average Minimum from all Tables.
	Degrees F.	Degrees F.	Degrees F.
2,250 station, C. & C.	67	73.3	62.7
2,200 station, Ophir winze	101.2	102.1	100.6
2,150 drift	99.2	101.6	95.3
2,250 drift	105.6	110.5	101.8
2,350 drift	116.9	125.4	108.5
2,100 stopes	101.2	103.0	99.0
2,200 stopes	103.0	103.2	102.8
2,200 raise	109.7	109.7	......

The temperatures given for the 2,250 and other stations of the C. & C. show the temperatures of the incoming air-currents on the different levels. The measurements in the drifts give the temperatures after the air has been heated to a maximum by contact with the walls of the drifts. A particularly hot zone was passed through by the 2,350 drift close to the 2,350 station of the C. & C. Before connection was made with the Ophir winze, the temperatures were so excessive as to cause the miners much suffering. As soon as a connection was made the temperature dropped about 15°. A progressive increase in temperature with depth is to be noted in the case of the drifts. Few measurements were made at different elevations in the stopes, but the air-current rises in temperature as it ascends in the stope. Confined mine-workings, such as drifts and raises, were usually observed to be the hottest workings, especially where no air-connections exist. Large mine-openings, such as stopes, are usually cooler.

III. The Ventilating-System in General.

A ventilating-current on the Comstock must provide pure air for the miners and a sufficient volume to temper the heat of the underground workings; it must also remove the steam and the gaseous products of blasting. A ventilating-current of sufficient volume to cool the workings enough to be noticeable would meet all the other requirements. The general and consider-

able elevation of temperature along the lode makes it possible to secure air-currents of some volume and velocity by natural ventilation alone. With the exception of the Ward shaft, all of the ventilation is effected by upcast shafts, supplemented by small fans for local distribution of the air in drifts, raises, stopes, and winzes. Three shafts, the Ophir, the Combination, and the Belcher, are used as upcasts; six shafts, the Union, the C. & C., the Mint, the old Yellow Jacket, the Overman, and the Alta, are used as down-casts. The Ward shaft is divided by a brattice; and one compartment (the pump) is used as an upcast, the other two compartments, in which the hoisting is done, serving as a down-cast. A fan is attached to the upcast compartment and exhausts the air.

1. *Ophir Upcast.*—This shaft takes the air from the workings of the Ophir mine, the down-cast air coming principally from the C. & C. shaft. The upcast air is taken from a winze in the north lateral and by a cross-cut to the Ophir incline, and thence to the foot of the Ophir shaft. At the foot of the Ophir shaft a south drift takes air from old stopes and workings of the Ophir mine, and an east cross-cut connects with the Union shaft and takes air from that shaft.

2. *Combination Upcast.*—This shaft draws air from the Sutro tunnel, both east and west of the Combination partition, which is so placed as to split the connection from the Combination shaft to the Sutro tunnel. The down-cast air comes from the Union, Mint, and Alta shafts. The air from the Union crosses the current from the Ophir through a 20-in. galvanized-iron pipe, two wooden partitions keeping the currents from mixing. The air from the Mint shaft is taken through three lines of 15-in. air-pipe, assisted by fans, to the Gould and Curry, the Savage, the Chollar, and the Hale and Norcross workings. It is then returned through the north and south laterals to the Sutro tunnel, and thence to the Combination shaft. The air from the Alta passes through a long connection and into the south lateral near the connection to the Ward shaft. The air east of the Combination partition comes from the Sutro-tunnel portal.

3. *Belcher Upcast.*—This shaft ventilates the Yellow Jacket, the Overman, and the Caledonia mines, and draws air for this purpose down the old Yellow Jacket and the Overman shaft.

The total amount of upcast air from the three shafts and the

Ward measures 235,835 cu. ft. per min., and the amount of down-cast air measures 216,687 cu. ft. per min. The discrepancy between the two amounts is due to the difficulty of making accurate measurement, of rise in temperature, leakage, variation in the velocity of the air-current at different times, and the accession of water-vapor and gases from the mine-workings. The distribution of the air between the shafts is shown by Table VI.

TABLE VI.—*Distribution of Air in Upcasts, Down-Casts, and the Sutro Tunnel.*

Shaft.	Number of Compartments.	Area.	Quantity.	Temperature at Collar.	Temperature at Level.	Connecting Level at Bottom.	
		Sq. Ft.	Cu. Ft. per Min.	Deg. F.	Deg. F.		
Combination	4	129	51,651	87.6	95.2	1,600	
Ophir	4	94	58,454	88.9	98.8	1,600	
Belcher	3	60	31,706	73.9	87.3	1,100	Yellow Jacket
Ward	1	33	94,024	67.5	104	2,475	
Total upcast air	12	316	235,835				
C. & C.	3	77.3	28,941				
Union	4	126	13,814				
Yellow Jacket	3	64.2	10,665				
Alta	2	43.3	13,364				
Mint	1½	30	7,500				
Overman on 900-ft. level	3	60	5,981				
Unaccounted for in Belcher	....	....	15,060				
Sutro tunnel	1	63	27,339				
Ward shaft	2	55	94,024				
Total down-cast air	19½	518.8	216,687				

From Table VI. the average velocity in feet per minute has been calculated as follows:

	Cu. Ft. Per Min.
Whole upcast area,	746.3
Whole down-cast area,	417.6
Whole upcast area, excluding Ward shaft,	501.1
Whole down-cast area, excluding Ward shaft,	264.4

An interesting comparison with past conditions is afforded by the data given by Church.[4] Table VII. is quoted from his treatise.

[4] *The Comstock Lode: Its Formation and History*, p. 18 (1879).

TABLE VII.—*Distribution of Upcast Air, July 2, 1877.*

Shaft.	Quantity.	Temperature of Up-cast at Top of Shaft.
	Cu. Ft. per Min.	Degrees F.
Utah	4,000	.....
Sierra Nevada	7,700	76
C. & C.	21,600	84
Con. Virginia	48,750	89
Gould and Curry	12,000	.....
Savage	58,500	100
Chollar-Potosi	18,000	77
Bullion	10,080	89.5
Imperial Consolidated	28,800	95
Belcher	52,200	89
Overman	27,000	93
Total upcast air	288,630	Aver. temp.....88.05

Six shafts served as down-casts, but were not named. Church estimates the above quantity of air as a minimum, and states that probably 300,000 cu. ft. represents the average amount of upcast air, and 10,000 cu. ft. per min. as the quantity of air due to the air-compressors in use. The outside temperature for the day upon which the measurements were made is given as 73° F.

The average amount of air per upcast, from Table VII., is 26,240 cu. ft. per min., and temperatures at the top of the shaft average 88.05°. Excluding the Ward shaft, Table VI. shows an average of 47,270 cu. ft. per min. per shaft, and a top-temperature average of 83.4°. The figures indicate a somewhat greater effect for the upcast shafts at the present time than in the past. The lower average temperature shows the effect of the restricted workings. The fewer upcast shafts and the restricted workings of the present time would indicate, on the whole, a greater resistance to the movement of the air than at the time Church made his studies. The greater effectiveness of the present upcast shafts could be explained by the more liberal use of small fans. No detailed data are available concerning the use of small fans during the early periods of the Comstock. Of the shafts mentioned by Church, only the C. & C., Belcher, and Overman are open, and of these the C. & C. and Overman are used as down-casts, while the Belcher is an upcast at the present time.

Fig. 3 shows the relation of the underground workings, the principal upcast and down-cast shafts.

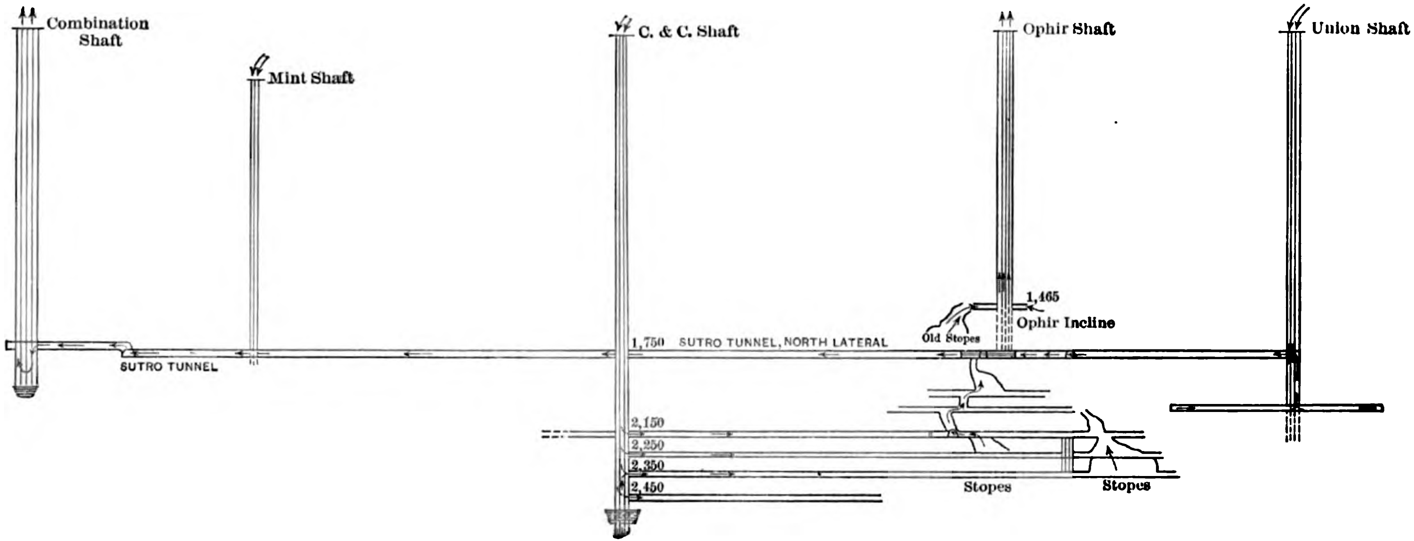

Fig. 3.—Longitudinal Section, Showing Workings and Shafts of North-End Mines.

IV. The Ventilating-System in Detail.

1. *Union Shaft.*—The down-cast air is taken to the 2,000-ft. level, and thence by means of a 15-in. air-pipe to a fan 600 ft. distant in the east cross-cut. It is then forced to the face of the NW. cross-cut, and also to the face of the south drift, distances respectively of 500 and 1,000 ft. The discharges are respectively 1,042 and 796 cu. ft. per min. The combined discharge is 1,838 cu. ft. per min. With an inflow at the Union shaft of 4,287 cu. ft. per min., the delivery is 42.9 per cent. of inflow. The air returns through the east cross-cut to the shaft and then passes up through the pump-compartment, which is bratticed off, to the Sutro-tunnel level, and then through a long connection to the south lateral and also to the connection leading to the Ophir shaft. The north lateral takes 4,086 cu. ft. per min., and the remainder, 9,728 cu. ft. per min., finds its way into the Ophir upcast. The average temperature on the 2,000-ft. level is 98.3° F., and on the Sutro-tunnel level it is 76.3°. This is due to the fact that a comparatively large proportion of the down-cast air leaks past the air-pipe on the 2,000-ft. level and goes up the pump-compartment, cooling the air coming from the Union workings. The portion of air passing to the Combination shaft traverses 12,061 ft. from surface to surface; the portion passing up the Ophir moves 7,680 ft. Fig. 4 shows in plan the distribution of the air on the Sutro-tunnel level, and Table VIII. gives the measurements made on the same level.

2. *The Ophir Mine.*—The down-cast air in the C. & C. shaft splits into four parts—one split on the 2,150-ft. level, an air-current of 7,024 cu. ft. per min.; one on the 2,250-ft. level, an air-current of 14,521; one on the 2,350-ft. level, of 11,667, and one on the 2,350-ft. level, which was not measured.

On the 2,150-ft. level the air is taken through the drift to a point 860 ft. from the shaft, then picked up by a fan and forced to the Ophir winze and down through that connection to the stopes upon the 2,200-ft. level of the Ophir. The fan takes 5,389 cu. ft. per min., and the difference, 1,635 cu. ft., leaks past a canvas curtain and joins the upcast air.

On the 2,250-ft. level part of the air is taken by a fan at the station and forced down to the 2,350-ft. level to ventilate the

TABLE VIII.—*Measurements of Air, Temperatures, and Relative Humidities at the Sutro-Tunnel Level.*

Section.	Date, 1908.	Velocity.	Area Section.	Quantity.	Temperature.	Relative Humidity.	Notes.
		Ft. per Min.	Sq. Ft.	Cu. Ft. per Min.	Deg. F.	Per Cent.	
F.	Dec. 28.	228	63	13,364	84.2	86.2	Air from Alta shaft.
F.	Dec. 18.	161	63	10,134	86.0	84.6	Air from Alta shaft.
G. G. & C.	Nov. 15.	1,159	1.23	1,426	65.2	71.0	To Gould and Curry through 15-in. pipe.
G. S.	Nov. 15.	904	2.66	2,404	65.2	71.0	To Savage through 15-in. pipe.
G. H. & N.	Nov. 15.	2,452	1.39	3,408	65.2	71.0	To Hale and Norcross through 15-in. pipe.
G. G. & C.	Dec. 28.	1,122	1.23	1,380	63.1	77.8	To Hale and Norcross through 15-in. pipe.
G. S.	Dec. 28.	886	2.66	2,357	63.1	77.8	To Hale and Norcross through 15-in. pipe.
G. H. & N.	Dec. 28.	2,706	1.39	3,761	63.1	77.8	To Hale and Norcross through 15-in. pipe.
T.	Dec. 18.	1,035	8.1	8,221	87.8	91.4	Through door.
Winze.	July 15.	980	20.0	19,600	107.6	100—	Mine air.
J.	July 16.	650	39.0	25,336	105.4	100—	Curtain at N. closed.
J.	July 16.	438	39.0	16,454	105.4	....	Curtain at N. open.
J.	July 17.	600	39.0	23,400	104	97	Curtain at N. closed.
J.	July 17.	329	39.0	12,831	104	....	Curtain at N. open.
J.	Oct. 11.	567	54.5	30,901	93.2	95	Partition at N.
J.	Dec. 12.	874	39.0	30,186	99.0	81.2	Partition at N.
J.	Dec. 29.	940	35.0	32,760	100.4	76.4	Partition at N.
K.	July 15.	1,060	29.7	31,482	101.8	100—	Curtain at N. closed.
K.	July 16.	1,108	29.7	32,907	100.4	100—	Curtain at N. closed.
K.	July 17.	730	35	25,550	98.6	97.9	Curtain at N. closed.
K.	July 17.	809	35	28,315	95.0	....	Curtain at N. open.
M.	July 16.	464	35	16,440	105.8	....	Air from old stopes.
M.	July 17.	330	35	11,550	104.9	....	Air from old stopes.
L.	July 16.	210	20	4,200	98.2	....	Air from Union shaft.
L.	July 17.	145	20	2,900	95.9	....	Air from Union shaft.
R.	July 15.	190	56	10,640	91.4	....	
R.	July 16.	190	35	•6,650	87.8	....	Curtain closed at N.
R.	July 16.	376	35	13,160	87.8	....	Curtain open at N.
R.	July 17.	120	35	4,200	88	73.4	Curtain closed at N.
R.	July 17.	221	35	7,735	85.8	....	Curtain open at N.
R.	Dec. 12.	3,291	0.4	1,316	86.4	66.8	Air-pipe in partition.
N.	Dec. 12.	364	22.4	8,148	82.4	72.4	Curtain removed.
O.	Dec. 12.	278	14.72	4,086	82.4	72.4	Air going to I.
P.	Dec. 12.	109	31.00	3,379	82.4	72.4	
Q.	Dec. 12.	478	2x.85	13,814	82.4	70.8	
H.	Oct. 11.	1,632	2.07	3,378	97	46.5	20-in. pipe.
H.	Dec. 29.	1,750	2.07	3,622	94	48.0	20-in. pipe.
I.	Oct. 11.	1,750	2.00	3,500	93.6	49.0	20-in. pipe.

east drift; the remainder, amounting to 14,521 cu. ft. per min., passes into the drift until it reaches a stope, where two fans pick it up, one forcing the air past the stope into the Consolidated Virginia workings, the other forcing the remaining air down through the Ophir winze to the 2,200-ft. level of the Ophir mine. The Consolidated Virginia fan (No. 3) takes 3,323 cu. ft. per min., while the Ophir fan (No. 2) takes 11,566 cu. ft. per min. Beyond the stope another fan (No. 1) takes air from the Ophir-winze station and forces it on to the stopes north of the Ophir winze. This air rises up through the stopes and event-

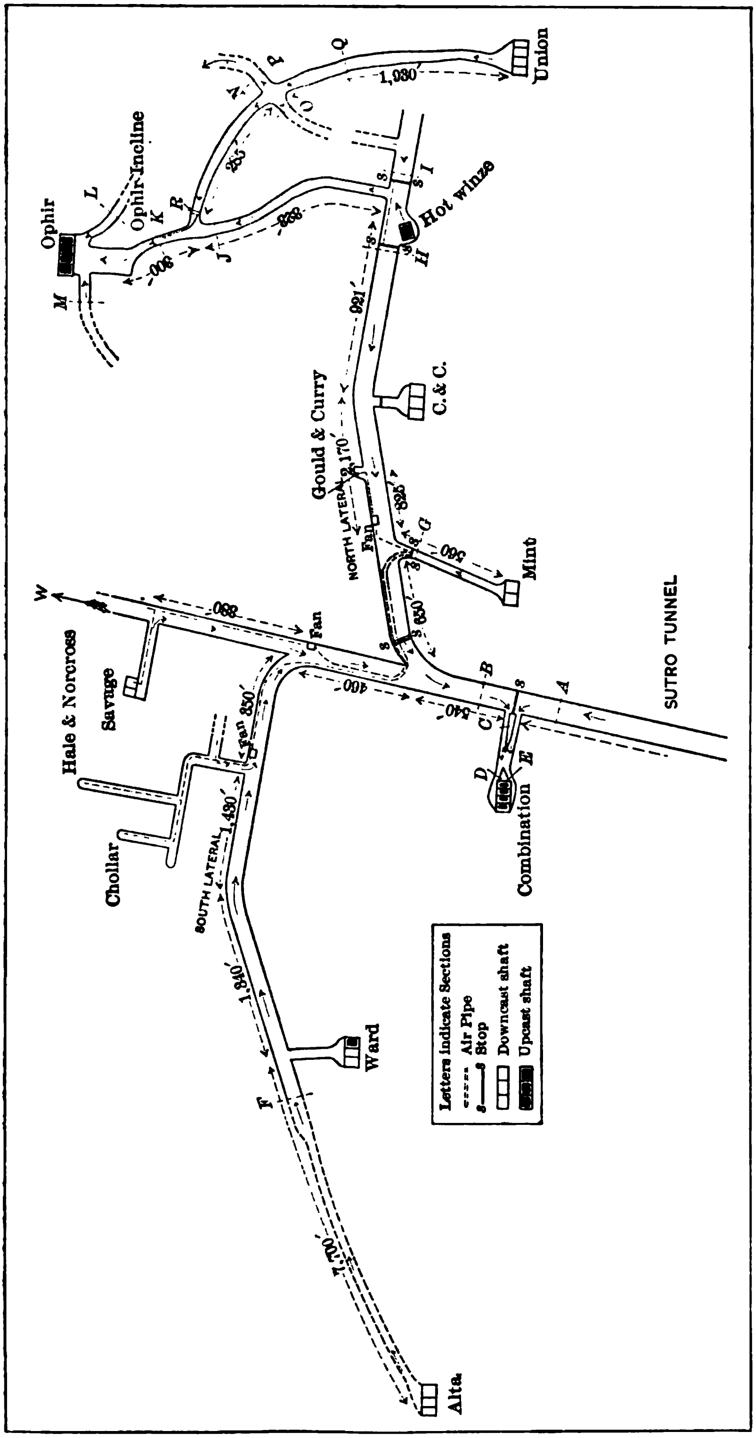

FIG. 4.—PLAN SHOWING DISTRIBUTION OF AIR ON SUTRO-TUNNEL LEVEL.

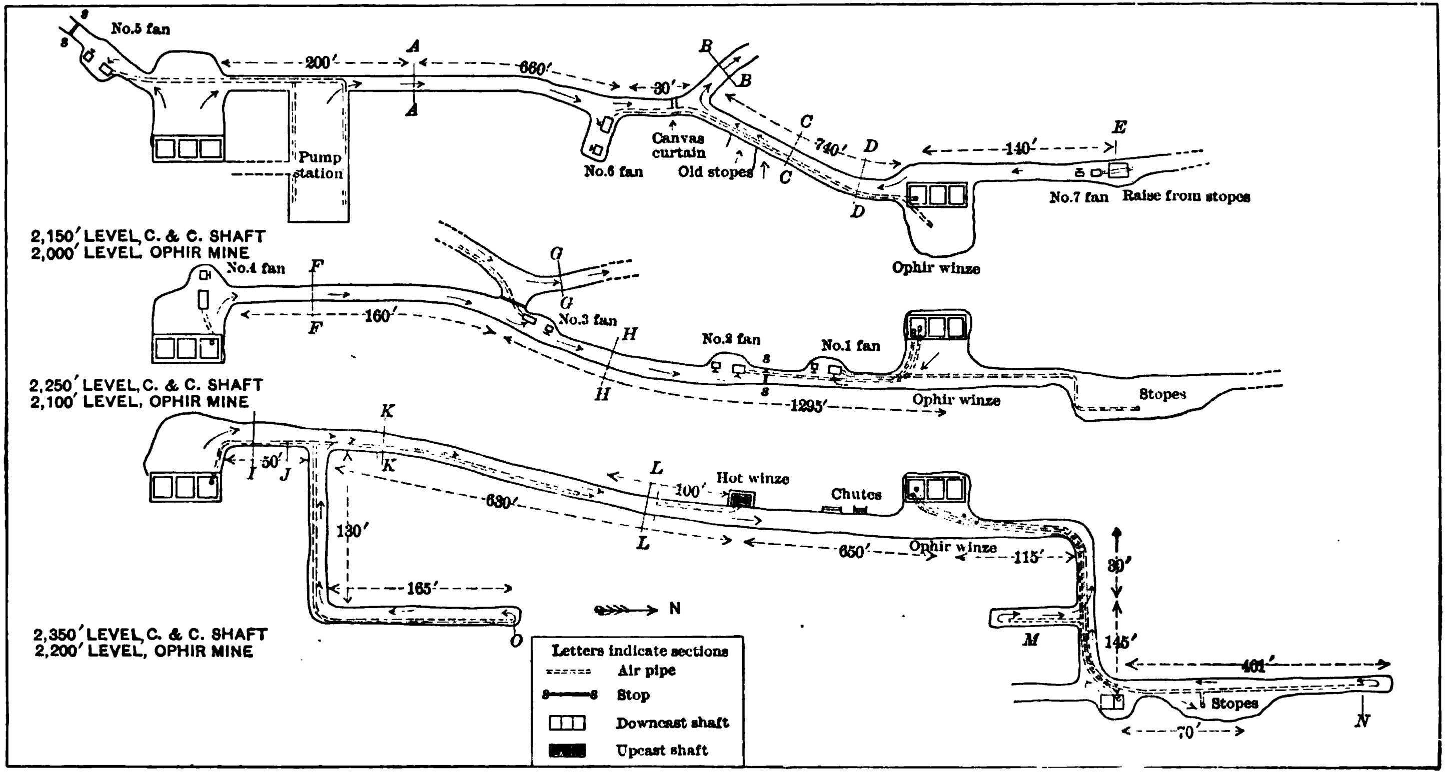

FIG. 5.—PLAN OF 2,000-, 2,100- AND 2,200-FT. LEVELS OF THE OPHIR MINE.

ually joins the air in the "hot winze" of the north lateral. The fan takes 3,791 cu. ft. per minute.

The air on the 2,350-ft. level passes through the main drift, while the air from the fan on the 2,250-ft. level passes into a cross-cut and a drift east of the main drift, returns, and joins the air passing through the main drift towards the Ophir winze. The high temperatures, noted previously, were taken in this drift before the connection was made. The average temperature before connection was 125.4°, and some months later the temperature measured 100.4°. The lowering in temperature was due to the increased volume of air and to the boarding-up (snow-shedding) of the sides and top of the drift, which very effectually shut off hot water, steam, and much of the heat radiated from the rock-surfaces.

The 2,450-ft. level is ventilated by a No. 8 cast-iron American blower. The air is taken through an inlet-pipe extending towards the south compartment of the shaft and forced to the working-face in the east cross-cut, returning through the cross-cut. The fan was not in operation at the time of my visit. On account of the danger of flooding this level the fan was driven by a compressed-air engine.

The total quantity of air in the three main splits amounts to 33,212 cu. ft. per min., and this compares closely to the 32,760 cu. ft. measured on the same day on the Sutro-tunnel level. Fig. 5 shows the plans of the three levels of the Ophir mine, with the sections at which velocity-measurements were made. The measurements are given in Table IX.

3. *Gould and Curry Mine.*—At the Sutro tunnel a west cross-cut has been started. This is ventilated by an air-pipe and fan which takes air from the Mint shaft. The inflowing air at the shaft measured 1,122 cu. ft. per min. (1,159 cu. ft., November 15). A 15-in. air-pipe takes this air to the fan, and from the fan an 11-in. pipe connects with the cross-cut, distant 189 ft. The air discharged measured 989 cu. ft., or 88.1 per cent. of the inflow. The fan was not in use.

4. *Savage Mine.*—An east cross-cut is being run in the Savage mine from a raise 75 ft. above the Sutro tunnel, which also receives air from the Mint shaft. A 15-in. air-pipe takes 2,404 cu. ft. per min. to a fan 1,110 ft. distant in the Sutro tunnel, and the fan discharges through a 15-in. pipe to the top of the

TABLE IX.—*Air-Measurements at Different Sections in Ophir Mine.*

Date, 1908.	Level.	Section.	Area.	Velocity.	Quantity.	Temperature.	Relative Humidity.	Notes.
			Sq.Ft.	Ft. per Min.	Cu. Ft. per Min.	Deg. F.	Per Cent.	
July 15.	2,000	A.	29.75	240	7,140	97.2	36.5	
Dec. 29.	2,000	A.	31.50	223	7,024	87.4	34.2	
July 15.	2,000	No. 6 fan.	1.77	2,766	4,896	....	......	Velocity low.
Dec. 29.	2,000	No. 6 fan.	1.77	3,045	5,389	98.6	37.5	Inlet of fan.
July 15.	2,000	B.	29.75	480	14,280	116.6	100—	Air to upraise to Sutro tunnel.
Dec. 29.	2,000	B.	23.22	620	14,396	108.1	86.7	Air to upraise to Sutro tunnel.
July 15.	2,000	C.	28.05	175	4,900	112.3	45.4	
Dec. 29.	2,000	C.	26.00	156	4,056	105.8	58.8	
Dec. 29.	2,000	D.	25.6	434	11,110	107.6	59.1	
July 15.	2,000	E.	20	225	3,375	100	98.5	
July 15.	2,100	F.	29.75	397	11,810	76.1	95.4	
Dec. 29.	2,100	F.	31.5	461	14,521	64.8	93.3	
July 15.	2,100	G.	26	320	8,320	96	89	
Dec. 29.	2,100	No. 3 fan.	2.82	1,182	3,323	....	......	
July 15.	2,100	H.	29.75	280	8,380	82.4	79.8	
Dec. 29.	2,100	H.	30.9	284	8,776	90.1	55.3	
Dec. 29.	2,100	No. 2 fan.	7.01	1,650	11,566	90.1	55.3	
Dec. 29.	2,100	No. 1 fan.	1.67	2,270	3,791	100.4	49.7	Temperature taken at Ophir-winze station.
July 15.	2,200	I.	28.25	220	6,140	89.6	96.4	Connection made.
July 15.	2,200	J.	1.22	3,492	4,160	82.4	......	End of 15-in. air-pipe; air from No. 4 fan.
Dec. 29.	2,200	K.	28.25	413	11,667	84.2	88.4	
Dec. 29.	2,200	L.	....	....	....	98.6	98.8	End of snow-sheds.
Dec. 29.	2,200	Hot winze	....	....	....	102.7	93.8	
Dec. 29.	2,200	....	....	......	....	104	98.8	50 ft. south of Ophir winze
Dec. 29.	2,200	Ophir winze.	....	....	....	100.4	45.8	
Dec. 29.	2,200	O.	1	1,996	1,996	89.6	40.6	Air from pipe, No. 4 fan.
Dec. 29.	2,200	M.	1.22	606	739	102.2	48.5	Air from pipe, No. 1 fan.
Dec. 29.	2,200	N.	1.11	490	544	104.5	36.5	Air from pipe, No. 1 fan.
Dec. 29.	2,200	....	....	....	....	98.6	72.6	Stopes between 2,100- and 2,200-ft. levels.

raise and then by an 11-in. pipe to the working-face, 180 ft. distant from the top of the raise. The discharge measures 894 cu. ft. per min., or 37.1 per cent. of the inflow. The section of 11-in. pipe caused the most leakage. The fan was in operation at the time.

5. *Hale and Norcross Mine.*—The Hale and Norcross receives air from the Mint shaft. A 15-in. air-pipe leads to the H. & N. connection in the south lateral, 1,460 ft., and from this point a fan takes the air and forces it to the H. & N. and Chollar crosscuts, distant 325 ft.; 11-in. branch-pipes take the air to the working-faces, 190 ft. to the H. & N. and 122 ft. to the Chollar. The H. & N. discharge measures 941 and the Chollar 934 cu. ft. per min. The combined discharge measures 1,875 and the inflow at the Mint 3,761 cu. ft. per min. The discharge is 49.8 per cent. of inflow. The measurements of both the Chollar and the H. & N. are given in Tables X. and XI. and Fig. 6.

TABLE X.—*Air-Measurements in Hale and Norcross and Chollar Mines.*

Point.	Velocity.	Area.	Quantity.	Temperature.	Relative Humidity.	Notes.
	Ft. per Min.	Sq. Ft.	Cu. Ft. per Min.	Deg. F.	Per Cent.	
Mint Shaft.	2,452	1.39	3,408	65.2	71	Nov. 15, 1908. Air going into H. & N. air-pipe.
Mint Shaft.	2,706	1.39	3,761	63.1	77.8	Dec. 28, 1908. Air going into H. & N. air-pipe.
B.	1,416	0.66	934	104.5	23.8	Nov. 15, discharge of 11-in. pipe.
B.	....	....	....	105.1	61.4	Dec. 28, discharge at J, 78 ft. from B.
J.	1,093	0.66	721	106.2	22.0	Dec. 28, discharge of 11-in. pipe.
C.	....	....	....	97.5	69.0	Nov. 15.
C.	....	....	....	104.0	51.2	Dec. 28.
E.	1,426	0.66	941	106.1	21.8	Nov. 15, discharge of 11-in. pipe.
E.	....	....	....	107.6	32.2	Face of drift 12 ft. from end of pipe, Nov. 15.
E.	1,043	0.66	688	109.4	20.0	Discharge of 11-in. pipe, Dec. 28.
E.	....	....	....	109.6	29.8	Face of drift 25 ft. from end of pipe, Dec. 28.
D.	....	....	....	95.5	66.2	Nov. 15.
F.	....	....	108	81 7	51.4	Leakage, 15-in. damper; cooling bench, Nov. 15.
F.	2,410	1.39	3,350	104.0	24.8	Damper on split closed; F open, Dec. 28.
F.	1,360	1.39	1,890	104 0	24.8	Both dampers open, Dec. 28.
F.	8	1.39	....	104.0	24.8	Damper at F closed, Dec. 28.
G.	9	....	....	98.1	64.4	Nov. 15.
G.	....	....	....	100.4	62.5	Dec. 28.
H.	....	....	....	105.8	66.6	Nov. 15.
H.	....	....	....	108.1	62.7	Dec. 28.
A.	....	....	....	105.5	58.5	Nov. 15.
A.	....	....	....	104.0	63.8	Dec. 13.
H.	....	....	....	108.86	55.9	Dec. 13.

TABLE XI.—*Inflow and Discharge of Air at Hale and Norcross and Chollar Mines.*

Date, 1908.	Inflow.	Discharge.	Air Lost.	Quantity Delivered.	Distance.
	Cu. Ft. per Min.	Cu. Ft. per Min.	Cu. Ft. per Min.	Per Cent.	Ft.
Nov. 15......	3,408	1,983	1,425	58	2,137
Dec. 28.......	3,761	1,409	2,352	37.4	2,155
Dec. 28.......	3,761	3,350	411	89	1,730

6. *Sutro Tunnel.*—The Sutro tunnel is an incast air-way and is the longest air-way on the Comstock. A plan of the tunnel at the Combination partition is shown in Fig. 7. The air enters at the portal and passes to the Combination partition,

a distance of 18,700 ft.; then through the connection to the Combination shaft, a distance of 400 ft., and then to the surface, a distance of 1,600 ft. vertically. The total distance is 20,700 ft. This Combination shaft also serves to draw air from the workings west of the Combination partition. The average quantity of air from the east is 27,339 cu. ft. per min. (average of four observations at different times); from the west it is 29,961 cu. ft. per min. (average of four observations at different times); and the total average passing to the Combina-

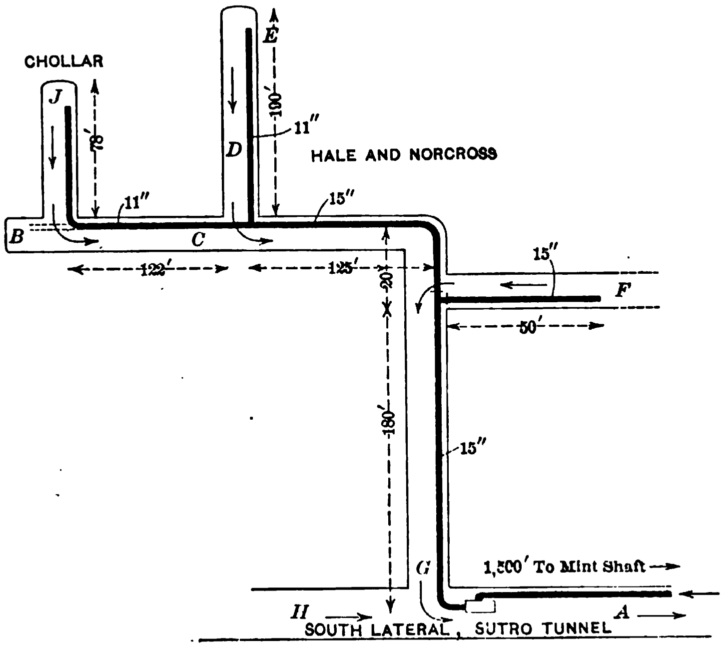

FIG. 6.—PLAN OF CHOLLAR AND HALE AND NORCROSS WORKINGS.

tion from both directions measures 57,300 cu. ft. per min. The observations made at the Combination partition are tabulated in Table XII.

An interesting comparison is afforded by a measurement made in the Sutro tunnel, Mar. 7, 1909. The Sutro tunnel just west of the Combination partition was completely blocked by a cave, and the Combination shaft drew air only from the east side. The quantity of air measured 30,124 cu. ft. per min.; temperature, 89.6° F.; relative humidity, 100 per cent. The temperature at the foot of the shaft measured 92.3°; relative

TABLE XII.—*Observations on Air-Currents at the Combination Partition.*

Date, 1908.	Section.	Velocity.	Area.	Quantity.	Temperature.	Relative Humidity.	Barometer.	Time.
		Ft. per Min.	Sq. Ft.	Cu. Ft. per Min.	Deg. F.	Per Cent.	In.	
Oct. 11..............	D.	730	35.72	26,075	94.3	96	....	11.20 a.m.
Oct. 11..............	E.	850	27.58	23,443	95.4	95.4	....	11.20 a.m.
Oct. 11..............	C.	1,154	18.9	21,810	95.54	85.7	....	11.20 a.m.
Oct. 11..............	A.	492	48.5	23,813	97.52	93.0	....	11.20 a.m.
Nov. 15............	A.	523	48.5	26,500	95.0	100	25.4	....
Nov. 22............	A.	631	48.5	32,181	94.5	98.4	25.15	9.00 a.m.
Dec. 13............	A.	604	48.5	30,200	93.7	96.4	25.10	10.00 a.m.
Dec. 28............	A.	480	48.5	24,000	94.1	96.4	25.35	11.00 a.m.
Nov. 15............	B.	627	46.3	29,307	95.5	85	25.4	
Nov. 22............	B.	732	46.3	33,891	95.2	83.9	25.1	9.00 a.m.
Dec. 13............	B.	628	46.3	29,076	94.64	83.9	25.15	11.10 a.m.
Dec. 28............	B.	594	46.3	27,572	94.50	83	25.39	11.00 a.m.

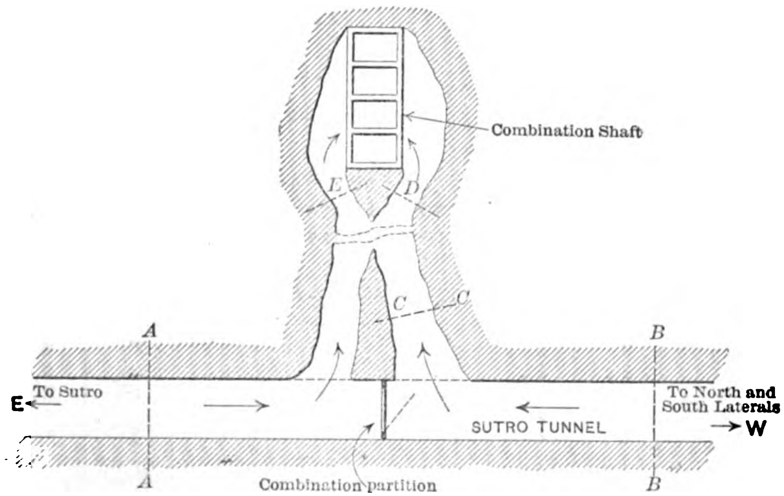

FIG. 7—PLAN OF SUTRO TUNNEL AT COMBINATION PARTITION.

humidity, 96.5 per cent.; barometer, 25.15 in. A measurement was taken on the same date, within 2 hr. of the measurement in the tunnel, at the collar of the Combination shaft, and 32,719 cu. ft. per min. at a temperature of 72°; relative humidity, 100 per cent.; and barometric height, 23.85 in., was the result. These figures show a pronounced cooling-effect of the shaft with a smaller volume of air passing, and this would de-

crease the efficiency of the shaft as an upcast. Instead of obtaining a greater volume of air by turning the full effect of the upcast shaft upon the tunnel, only a small increase resulted. This is a direct result of the decreased efficiency of the shaft.

A recording-anemometer was placed in the Sutro tunnel close to the Combination partition, and three 24-hr. records taken of the east air. The observed velocities are tabulated in Table XIII., the average velocities in Table XIV., and the quantities of air in Table XV. The anemometer was placed in the position shown in Fig. 8. The ratio between the average velocity for the section and the velocity for the position was determined by measurements with the small anemometer. This ratio was found to be 1.12, and this figure was used in reducing the observed velocities. The area of the section is 48.5 sq. feet.

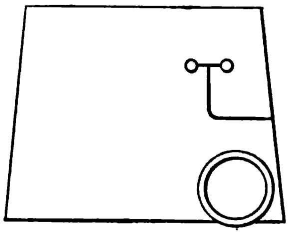

FIG. 8.—SECTION OF SUTRO TUNNEL EAST OF COMBINATION PARTITION, SHOWING POSITION OF ANEMOMETER.

TABLE XIII.—*Velocity-Readings.*

Time.	Nov. 21 to 22, 1908.	Nov. 22 to 23, 1908.	Nov. 23 to 24, 1908.	Time.	Nov. 21 to 22, 1908.	Nov. 22 to 23, 1908.	Nov. 23 to 24, 1908.
	Miles Per Hr.	Miles Per Hr.	Miles Per Hr.		Miles Per Hr.	Miles Per Hr.	Miles Per Hr.
12 to 1 p.m.	....	....	....	1 to 2 a.m.	5.00	4.70	5.60
1 to 2 p.m.	....	3.90	....	2 to 3 a.m.	5.00	4.80	5.50
2 to 3 p.m.	....	3.75	....	3 to 4 a.m.	4.20	4.62	5.90
3 to 4 p.m.	4.25	4.40	4.90	4 to 5 a.m.	4.34	4.75	5.50
4 to 5 p.m.	4.00	4.30	5.40	5 to 6 a.m.	3.90	5.13	5.50
5 to 6 p.m.	4.33	4.20	5.20	6 to 7 a.m.	4.16	4.80	5.40
6 to 7 p.m.	4.30	4.50	5.00	7 to 8 a.m.	4.55	4.80	5.74
7 to 8 p.m.	4.30	4.44	5.00	8 to 9 a.m.	4.62	5.00	5.45
8 to 9 p.m.	4.30	4.64	5.20	9 to 10 a.m.	4.60	4.80	5.40
9 to 10 p.m.	4.47	4.34	5.25	10 to 11 a.m.	4.26	4.20	5.30
10 to 11 p.m.	4.93	4.50	5.55	11 to 12 a.m.	4.50	4.10	5.30
11 to 12 p.m.	5.07	4.55	5.64	12 to 1 p.m.	4.25	4.20	4.60
12 to 1 a.m.	5.13	5.10	5.38	1 to 2 p.m.	....	4.00	....

TABLE XIV.—*Average Velocities.*

Date, Nov., 1908.	Average.		Maximum.		Minimum.		Time, Maximum.	Time, Minimum.
	Miles Per Hr.	Ft. Per Min.	Miles Per Hr.	Ft. Per Min.	Miles Per Hr.	Ft. Per Min.		
21 to 22.	4.98	438.3	5.75	505.2	4.37	384.4	12 to 1 a.m.	5 to 6 a.m.
22 to 23.	4.65	409.2	5.75	505.2	4.37	384.4	5 to 6 a.m.	1 to 2 a.m.
23 to 24.	5.99	527.1	6.61	581.3	5.16	453.4	3 to 4 a.m.	12 to 1 p.m.
Average...	5.21	458.5	6.04	530.6	4.63	407.4	....	....

The observed velocities have been plotted in Fig. 9.

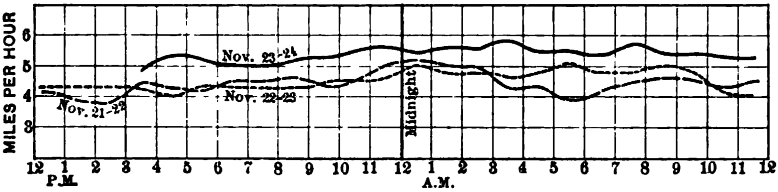

FIG. 9.—CURVES SHOWING RELATION BETWEEN VELOCITY AND TIME OF DAY.

TABLE XV.—*Air in Cubic Feet Per Minute.*

Date.	Average Quantity of Air.	Maximum Quantity of Air.	Minimum Quantity of Air.
	Cubic Feet.	Cubic Feet.	Cubic Feet.
Nov. 21 to 22....	21,257	24,502	18,859
Nov. 22 to 23....	19,846	24,502	18,859
Nov. 23 to 24....	25,564	28,193	21,990
Average...........	22,222	25,732	19,903

The average quantity, as shown in Table XV., and all the velocities, with the exception of one, are lower than the average velocity obtained from the readings in Table XII. Maximum and minimum velocities do not occur at any particular time. The wavy character of the velocity-curves is noteworthy, and is probably due to the fact that at several points in the tunnel are constrictions, often for some little distance, and the added friction produced by these results in more or less of a pumping-action. Undoubtedly, a more delicate recording-instrument would have shown the larger waves to be formed of a multitude of smaller ones. The movement of cars in the tunnel does not appear to have any decided effect upon the curves, although more delicate measuring-instruments might have shown it.

7. *Yellow Jacket Mine.*—Three levels of this mine are open

and operated from the old Yellow Jacket shaft. On the first level, the 900-ft., the air entering measures 3,075 cu. ft. per min. The air passes through a drift and thence down a winze to the 1,000-ft. level. The temperature and relative humidity of this air-current, at a point 500 ft. from the shaft, measured 66.2° F. and 55.6 per cent. respectively. The second, or 1,000-ft. level, receives 3,030 cu. ft. per min. from the shaft. Part of this air is taken by an air-pipe and conducted a distance of 350 ft. to a raise on the foot-wall of the vein, returns, joins the remainder, and then passes down through old stopes to the 1,100-ft. level.

The third, or 1,100-ft. level, is closed by a door in the drift, so as to force the air down the incline, which heads at this level, into an old station which is being reopened. The air escapes up through old stopes to the 1,100-ft. level. All the air passing down the Yellow Jacket shaft is brought together in the south drift, passes through a cross-cut to the old Belcher incline, and thence to the surface through the Belcher shaft. The amount of air passing through the drift measures 10,665 cu. ft. per min. The 900-ft. level of the Overman and Caledonia, corresponding to the 1,100-ft. of the Yellow Jacket, delivers 5,981 cu. ft. per min. to the Belcher shaft. The total quantity of air measured on this level is 16,646 cu. ft. per min., while the average passing up the Belcher is 31,706, leaving unaccounted for 15,060 cu. ft. per min. This comes from the lower levels of the Overman and Caledonia, but no measurement was possible at the time. The Yellow Jacket air at the foot of the Belcher incline gave a temperature of 79.7° and a relative humidity of 68 per cent.; the air coming from the Overman a temperature of 82.4° and a relative humidity of 62.3 per cent.

8. *Ophir and Belcher Upcasts.*—The measurements of both these shafts are to be found in Tables XVI. and XVII.

9. *Combination Shaft.*—The measurements for the Combination shaft are given in detail in Table XVIII. The numbered circles in Fig. 10 indicate the position in which the anemometer was held for each measurement.

Table XIX. has been calculated from the data of Tables XVI., XVII., and XVIII.

Since the cooling of the upcast air directly affects the effi-

TABLE XVI.—*Air-Measurements in Ophir Upcast Shaft.*

Date, 1908.	Pump.	No. 1.	No. 2.	No. 3.	Average.	Quantity.	Shaft-Temperature.	Relative Humidity, Shaft.	Surface-Temperature.	Relative Humidity, Surface.	Barometer.
						Cu. Ft. per Min.	Deg. F.		Deg. F.		
July 14, 11.45 a.m.	647	699	686	....	660	62,040	90.5	100+	73.4	30.45	24.2-
July 17, 1 p.m......	532	506	480	....	506	47,564	......	100+	81.0	16.40	24.2
Aug. 8, 4 p.m.......	546	588	506	....	523	49,162	93.4	100+	87.8	18.40	24.0
Oct. 18, 2 p.m......	726	405	759	....	630	59,220	89	100+	40.0	....	23.8
Nov. 21, 4.30 p.m.	716	701	747	....	721	67,774	89	100+	45.0	61.2	23.7
Dec. 22, 5.30 p.m.	584	668	706	....	653	61,382	86	100+	40.6	26.5	23.97
Dec. 27, 3.30 p.m.	658	652	675	....	660	62,040	86	100+	43.3	50.1	24.0
Average............	....	....	....	....	622	58,484	88.9	100+			

TABLE XVII.—*Air-Measurements in Belcher Upcast Shaft.*

Date, 1908.	No. 1.	No. 2.	No. 3.	Average.	Quantity.	Shaft-Temperature.	Relative Humidity, Shaft.	Surface-Temperature.	Relative Humidity, Surface.	Barometer.
					Cu. Ft. per Min.	Deg. F.		Deg. F.		
Oct. 10, 4 p.m......	620	567	323	595	27,720	......	100+	......	....	......
Nov. 8, 4 p.m......	656	612	525	598	30,611	......	100+	......	....	......
Nov. 15, 10.45 a.m.	760	717	525	667	36,375	74.3	100+	50	....	......
Nov. 22, 9 a.m.....	835	724	405	655	33,915	74.0	100+	50	....	......
Dec. 18, 3 p.m......	578	541	586	561	28,714	73	100+	26.6	49.2	24.2
Dec. 30, 11.45 a.m.	692	613	323	543	32,992	74.3	100+	44.6	50.2	23.9
Average............	....	....	....	603	31,706	73.9				

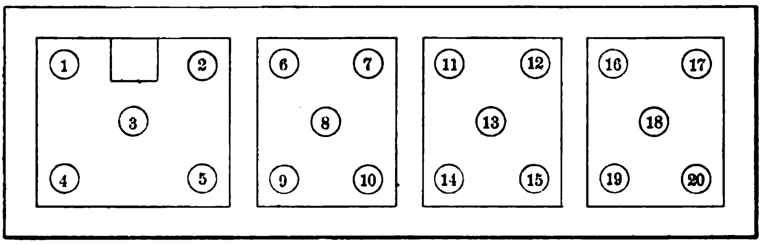

FIG. 10.—SECTION OF COMBINATION SHAFT, SHOWING POSITION OF ANEMOMETER.

ciency of the upcast shaft, the factors contributing to this are of some importance. I reached the following conclusions:

1. The greater the difference between the temperature of the upcast air and the rock-temperature, the greater the cooling-effect of the shaft.

2. The higher the temperature of the air entering the bottom of the upcast, the greater the cooling-effect of the shaft.

TABLE XVIII.—*Air-Measurements in Combination Shaft.*

Velocity, Feet per Minute.

Date, 1908.	Pump.						No. 1.					
	1.	2.	3.	4.	5.	Average.	6.	7.	8.	9.	10.	Average.
July 16....	110	315	330	200	405	272	312	400	312	350	570	389
Oct. 10....	180	260	300	370	420	306	440	300	.....	.....	.....	488
Nov. 14...	560	500	452	500	510	504	630	520	.....	622	270	510
Nov. 15..	.....	.....	.....	.....	.....	353	930	481	.....	630	363	601
Nov. 21...	436	453	459	445	490	456	612	488	.....	622	402	525
Dec. 27....	206	304	320	320	222	274	518	438	.....	526	286	442

Date, 1908.	No. 2.						No. 3.					
	11.	12.	13.	14.	15.	Average.	16.	17.	18.	19.	20.	Average.
July 16....	520	465	560	340	380	458	450	510	560	330	275	425
Oct. 10....	.....	.....	.....	.....	.....	411	350	370	600	410	422	430
Nov. 14...	510	515	.....	580	418	494	520	590	620	406	615	549
Nov. 15...	540	470	.....	310	400	430	464	600	600	390	560	523
Nov. 21...	.....	.....	.....	.....	.....	556	479	617	638	490	584	561
Dec. 27....	548	374	465	367	272	345	461	429	485	312	403	418

Cubic Feet of Air per Minute.

Date, 1908.	Pump C.	No. 1.	No. 2.	No. 3.	Total.	Average Velocity.
July 16....	10,472	11,670	13,590	13,750	49,482	385
Oct. 10....	11,781	14,640	8,220	12,900	47,551	434
Nov. 14...	19,404	12,240	9,880	16,470	57,994	514
Nov. 15...	13,590	14,424	8,600	15,690	52,309	476
Nov. 21...	17,556	12,600	11,120	16,830	58,111	524
Dec. 27....	10,549	10,608	10,350	12,540	44,047	370
Average..	13,892	12,697	10,293	14,696	51,582	450

Date, 1908.	Time.	Temperature, Shaft-Collar.	Relative Humidity, Shaft.	Surface-Temperature.	Relative Humidity, Surface.	Barometer.
		Deg. F.	Per Cent.		Per Cent.	In.
July 16....	10.00 a.m.	80.6?	100+	65.3	29.9	24.85
Oct. 10....	....	......	100+	......	......	....
Nov. 14...	3.30 p.m.	88	100+	50.0	......	23.92
Nov. 15...	3 to 4 p.m.	89	100+	52.9	33.8	23.90
Nov. 21...	3.00 p.m.	89	100+	42.8	65.0	23.70
Dec. 27....	1.30 a.m.	84.2	100+	46.0	54.0	24.00
Nov. 16...	2.00 p.m.	87.0	100+	61.0	......	24.00
Nov. 17...	3.00 p.m.	88.0	100+	64.0	......	24.00
Nov. 18...	3.00 p.m.	88.0	100+	55.0	......	23.80

TABLE XIX.—*Relation Between Depth of Shaft and Cooling of Air-Current.*

Shaft.	Temperature at Foot.	Temperature at Collar.	Depth of Shaft.	Loss in Temperature.	Loss in Temperature, per 1,000 Ft. Depth.	Loss in Temperature, per 1,000 Cu. Ft. Air.	Loss in Temperature, per 1,000 Sq. Ft. Rubbing-Surface Per Min.
	Deg. F.	Deg. F.	Ft.	Deg. F.	Deg. F.	Deg. F.	Deg. F.
Ophir	102.1	88.9	1,600	13.2	8.25	0.22	0.06
Belcher........	81.0	73.9	1,200	7.1	6.00	0.22	0.07
Combination.	95.1	87.6	1,600	7.5	4.70	0.145	0.02

Average temperature of upcast air, { Ophir, 95.5°. Belcher, 77.4°. Combination, 91.35°.

3. The lower the velocity, and consequently the smaller the volume of air in the upcast shaft, with given area and rubbing-surface, the greater the cooling-effect of the shaft.

4. The greater the proportion of rubbing-surface to shaft-area, the greater the cooling-effect.

5. Where incast and upcast air are conducted through the same shaft, the cooling-effect of the incast upon the upcast air is very marked.

Regarding the effect of humidity upon the cooling-effect of the shaft, no definite conclusion was reached.

The recording-anemometer used in the Sutro tunnel was placed in the Ophir shaft, but no very satisfactory results were obtained on account of the failure of the instrument to operate in a vertical position. A special fan to operate in a horizontal position was constructed and connected with the recording part of the instrument by means of bevel-gears. The instrument was placed in the pump-compartment of the Combination shaft, calibrated in position by means of small anemometers, and three records taken. The results are given in Tables XX., XXI., and XXII. The factor 0.84 was used in reducing the velocities as given by the instrument to average velocities for the shaft. This factor was determined by taking the average values for the ratio of average velocity and center velocity.

The results from Table XX. are plotted in Fig. 11. The curves show the same characteristics as those of the Sutro tunnel, with the difference that the irregularities are more accentuated. The "pumping-action" of the shaft, due to constrictions, is clearly shown. The shaft is of uniform section, but the air-way leading to the bottom narrows down to about two-thirds the area of both branches of the Sutro tunnel on either side of the Combination partition.

Comparing Tables XVIII. and XXII., the average quantity of air passing is greater by 9,513 cu. ft. per min. for the 24-hr. measurements than for the average of the single measurements. The average minimum 24-hr. measurement compares closely with the average of the results shown by the single measurements. This result is to be expected, for the reason that the afternoon marks the period of the minimum velocities, and most of the single measurements were made in the afternoon.

TABLE XX.—*Air-Velocity Measurements, Combination Shaft.*

Observed Velocities, Miles per Hour.

Time.	Nov. 14 to 15.	Nov. 16 to 17.	Nov. 17 to 18.
4 to 5 p.m.	6.20	5.35	5.60
5 to 6 p.m.	5.50	5.50	5.35
6 to 7 p.m.	6.70	6.10	5.60
7 to 8 p.m.	6.70	6.30	5.80
8 to 9 p.m.	6.30	6.30	5.80
9 to 10 p.m.	6.70	6.30	5.95
10 to 11 p.m.	6.30	6.30	6.10
11 to 12 p.m.	5.80	6.30	5.60
12 to 1 a.m.	6.30	6.10	6.70
1 to 2 a.m.	6.70	6.70	6.30
2 to 3 a.m.	6.70	6.30	6.20
3 to 4 a.m.	6.80	......	6.30
4 to 5 a.m.	6.70	5.80	6.70
5 to 6 a.m.	......	6.70	5.80
6 to 7 a.m.	......	6.70	6.80
7 to 8 a.m.	6.30	6.10	6.10
8 to 9 a.m.	6.10	5.80	5.60
9 to 10 a.m.	5.80	5.80	5.80
10 to 11 a.m.	5.80	5.35	5.80
11 to 12 a.m.	5.80	5.80	5.35
12 to 1 p.m.	5.60	5.35	5.80
1 to 2 p.m.	4.80	5.35	5.35
2 to 3 p.m.	......	5.35	5.10
3 to 4 p.m.	......	6.10	5.50

TABLE XXI.—*Average Velocities Over Shaft Section.*

Date, 1908.	Average Velocity.		Maximum Velocity.		Minimum Velocity.		Time of Maximum Velocity.	Time of Minimum Velocity.
	Miles per Hour.	Feet per Minute.	Miles per Hour.	Feet per Minute.	Miles per Hour.	Feet per Minute.		
Nov. 15 to 16...	5.67	547.1	5.71	549.7	4.03	388.8	3 to 4 a.m.	1 to 2 p.m.
Nov. 16 to 17...	5.03	485.3	5.58	583.6	4.49	433.3	1 to 2 a.m.	12 to 3 p.m.
Nov. 17 to 18...	4.87	469.9	5.71	549.7	4.28	413.9	6 to 7 a.m.	2 to 3 p.m.
Average..........	5.19	500.7	5.65	544.3	4.26	412.0	1 to 7 a.m.	1 to 3 p.m.

TABLE XXII.—*Volume of Air in Cubic Feet per Minute.*

Date, 1908.	Average.	Maximum.	Minimum.	Shaft-Temperature.	Shaft-Humidity.	Surface-Temperature.	Barometer.	Time.
				Deg. F.	Per Cent.	Deg. F.	In.	
Nov. 15 to 16......	66,745	67,063	47,441	89	100+	52.9	23.9	Nov. 15, 3 p.m.
Nov. 16 to 17......	59,210	65,098	52,784	87	100+	61.0	24.0	Nov. 16, 2 p.m.
Nov. 17 to 18......	57,390	67,063	50,498	88	100+	64	24.0	Nov. 17, 3 p.m.
				88	100+	55	23.8	Nov. 18, 3 p.m.
Average..............	61,095	66,408	50,224	88	100+	58.5	23.9	

10. *Supplementary Ventilation.*—At the Ward shaft a No. 200 American, three-quarter housing, vertical-discharge, suction fan, driven by a 50-h.p. induction motor, is used to ventilate the workings of the shaft; and, as sinking is the only work, with the exception of the pump-stations, practically the full capacity of the fan is used for this work. The incast air passes down the two hoisting-compartments, and is deflected in part by a canvas curtain at the 2,475-ft. station (pump-station). The curtain is one-half the width of the compartment, and is suspended by its upper edge from a wall-plate on a level with the roof of the station. The lower edge hangs loose over the guard-bar in front of the compartment. This gives a curved surface

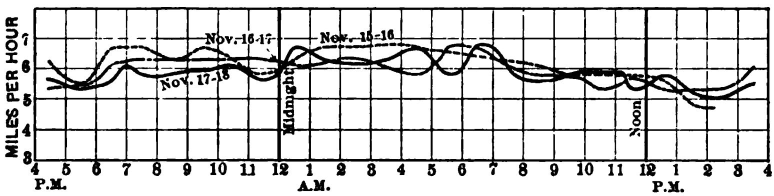

FIG. 11.—CURVES SHOWING RELATION BETWEEN OBSERVED VELOCITIES AND TIME OF DAY, COMBINATION SHAFT.

and deflects enough air to ventilate the station, while allowing the cage, in passing down, to sweep it out of the way. The greater part of the air passes down to the sump and then rises through the pump-compartment, which is bratticed off from the hoisting-compartments.

The discharge of the fan was measured several times. Five points were taken in the discharge section, and the average of the anemometer-readings was taken as the discharge velocity. Table XXIII. gives the results.

On December 13, at the 2,475-ft. level, the temperature in the station close to the down-cast compartments was 91.4° F., and the relative humidity 93.6 per cent.; in the upcast compartment on the same level the temperature was 104° and the relative humidity 100 per cent. plus. The upcast air was heavily charged with vapor.

The ventilating-effect of the fan is supplemented by the chimney-effect of the shaft. Calculating this effect, and not allowing for leakage through the brattice, a suction of 1.93 in. of water was obtained. This would leave 2.07 in. as the effect of the

TABLE XXIII.—*Air-Measurements of the Ward-Shaft Fan.*

Date, 1906.	1.	2.	3.	4.	5.	Mean Velocity.	Cu. Ft. Per Min.
July 19.....	2,842	4,200	3,556	2,842	3,576	3,476	105,280
Aug. 8......	2,248	3,860	3,464	2,280	3,292	3,028	91,597
Nov. 22....	2,760	3,428	3,044	2,628	2,944	2,959	89,510
Dec. 13.....	2,360	3,552	3,272	2,552	3,272	3,002	90,810
Mean	2,456	3,613	3,260	2,486	3,169	2,996	90,639

NOTE.—Only the last three readings were averaged, as the anemometer was not calibrated for the first reading.

Date.	Fan Discharge.		Air.		Time.
	Temperature.	Relative Humidity.	Temperature.	Relative Humidity.	
	Deg. F.	Per Cent.	Deg. F.	Per Cent.	
July 19......	......	......	......	......	9.00 a.m.
Aug. 8.......	90.0	37.0	88.34	14.0	9.00 a.m.
Nov. 22......	59.0	90.4	36.7	74.9	7.30 a.m.
Dec. 13......	55.4	94.3	36.5	56.5	12.50 a.m.

Water-gauge on suction side of fan gave 4 in. of water.

fan. The catalogue-rating of the fan is given as 75,700 cu. ft. per min. at 189 rev. per min. and an expenditure of 53.3 h.p. This would give, by difference, 14,939 cu. ft. per min. as due to chimney-draft. Calculation shows the chimney-effect to be equivalent to 17,088 cu. ft. per min. The difference between these two results is undoubtedly due to the catalogue-rating of the fan, which can be taken only in a very general way.

At the Ophir shaft a B. F. Sturtevant No. 16, double width, multi-vane fan is being installed. This fan is rated to deliver 140,000 cu. ft. per min. against a maintained suction of 2 in. of water at a speed of 200 rev. per min. with an expenditure of 100 horse-power.

The plan of the fan-arrangements at the Ward shaft is shown by Fig. 12 and at the Ophir by Fig. 13.

The distribution of the fans underground is given in Figs. 4 and 5, and the details of the fans in Table XXIV. The total horse-power of all motors used for ventilating purposes is 315; of this 150 is used on the surface and 165 underground. The actual horse-power in use, excluding the Ophir fan, which is being installed, is approximately 163.2. Of this 54 is used on

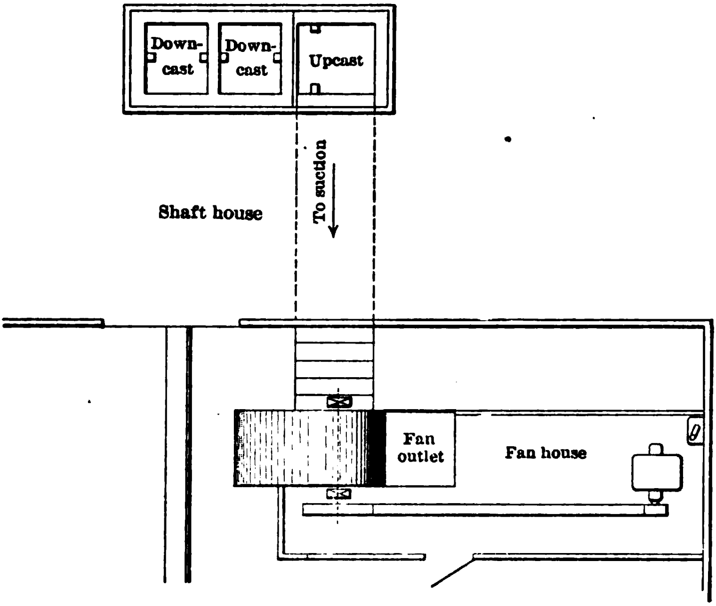

FIG. 12.—PLAN OF FAN AT WARD SHAFT.

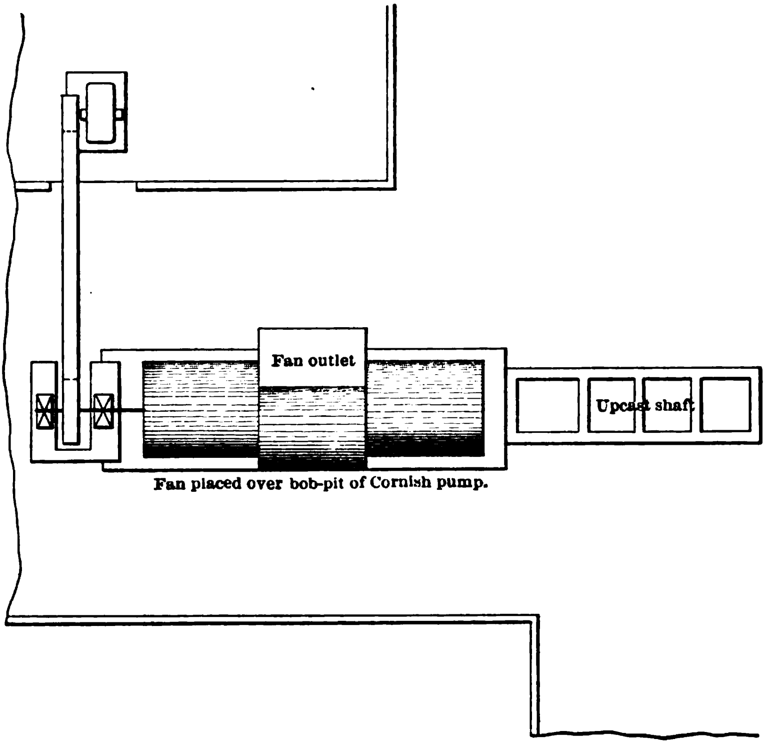

FIG. 13.—PLAN OF FAN AT OPHIR UPCAST.

TABLE XXIV.—*Fans in Use on the Comstock.*

Position.	Fan.	Motor.	Horse-Power.	Speed.	Air.	Diameter of Inlet.	Area of Inlet.	Diameter of Outlet.	Area of Outlet.	Diameter of Fan-Wheel.	Width of Fan-Wheel.	Notes.
		H.P.		Rev. per Min.	Cu. Ft. Per Min.	In.		In.				
Ward shaft	No. 200 metal exhauster	50	....	200	90,639	76	31.5 sq. ft.	....	31.25 sq. ft.	10 ft.	47 in.	
Ophir shaft	No. 16 double mine-fan	100	....	200	140,000	70	53.4 sq. ft.	....	58.8 sq. ft.	6 ft. 8 in.	7 ft. 4 in.	
Hale and Norcross, Sutro tunnel	15-in. metal exhauster	10	6	1,400	3,761	15	176 sq. in.	15	162 sq. in.	22 in.	12.75 in.	
Savage mine, Sutro tunnel	15-in. metal exhauster	10	6	1,000	2,357	15	176 sq. in.	15	162 sq. in.	22 in.	12.75 in.	
Gould and Curry, Sutro tunnel	15-in. metal exhauster	10	6	1,000	....	15	176 sq. in.	15	162 sq. in.	22 in.	12.75 in.	Not in operation.
Ward pump-station, 2,475 level	15-in. metal exhauster	5	2	....	....	15	176 sq. in.	15	162 sq. in.	22 in.	12.75 in.	
Union, 2,000 level	15-in. metal exhauster	10	8	1,680	4,287	15	176 sq. in.	15	162 sq. in.	22 in.	12.75 in.	Suction and exhaust.
Ophir, No. 1 fan	15-in. metal exhauster	15	4	1,000	8,791	15	176 sq. in.	15	162 sq. in.	22 in.	12.75 in.	
Ophir, No. 2 fan	6-ft. wooden fan	20	24	515	11,566	29	1,009 sq. in.	20 by 20	400 sq. in.	72 in.	20 in.	
Ophir, No. 3 fan	4-ft. wooden fan	5	4	360	3,828	20	406 sq. in.	16 by 16	256 sq. in.	52 in.	16 in.	
Ophir, No. 4 fan	6-ft. wooden fan	15	9.6	340	....	29	1,009 sq. in.	20 by 20	400 sq. in.	72 in.	20 in.	
Ophir, No. 5 fan	6-ft. wooden fan	20	16	500	....	29	1,009 sq. in.	20 by 20	400 sq. in.	72 in.	20 in.	
Ophir, No. 6 fan	15-in. metal exhauster	20	14.4	1,500	5,389	15	176 sq. in.	15	162 sq. in.	22 in.	12.75 in.	
Ophir, No. 7 fan	15-in. metal exhauster	....	....	....	....	15	176 sq. in.	15	162 sq. in.	22 in.	12.75 in.	Not in operation.
2,150 Ophir, north workings	4-ft. wooden fan	5	1.2	360	....	20	406 sq. in.	16 by 16	256 sq. in.	52 in.	15 in.	
1,000 level, Yellow Jacket	15-in. metal exhauster	10	....	1,000	....	15	176 sq. in.	15	162 sq. in.	22 in.	12.75 in.	Not in operation.
Overman-Caledonia	15-in. metal exhauster	10	....	1,680	4,900	16	201 sq. in.	16	201 sq. in.			

TABLE XXV.—*Accession of Moisture, Incoming Air-Currents.*

Nov. 22, 1908.	Temperature.	Absolute Humidity.	Increase Humidity.	Distance.	Exposed Surface.	Volume.	Velocity.	Area.	Perimeter.	Area / Perimeter.	Grains per Square Foot per Hour.	Grams per Square Meter per Hour.
	Deg. F.	Gr. per Cu. Ft.	Gr. per Cu. Ft.	Ft.	Sq. Ft.	Cu. Ft. per Min.	Ft. per Min.	Sq. Ft.	Ft.			
Sutro Tunnel.												
Portal	46	2.08										
0-5,200	71.6	6.21	4.13	5,200	182,000	32,181	450	71.4	35	2.04	48.8	30.553
5,200-6,800	73	7.29	1.08	1,600	56,000	32,181	450	71.4	35	2.04	37.2	25.949
6,800-8,000	85.2	11.79	4.50	1,200	42,000	32,181	450	71.4	35	2.04	206.9	144.326
8,000-10,000	91.9	15.23	3.44	2,000	70,000	32,181	450	71.4	35	2.04	94.9	66.200
10,000-13,000	95	16.05	0.82	3,000	105,000	32,181	450	71.4	35	2.04	15.1	10.533
13,000-17,000	95	16.64	0.59	4,000	140,000	32,181	450	71.4	35	2.04	8.1	5.650
17,000-18,000	94.4	16.99	0.35	1,000	35,000	32,181	450	71.4	35	2.04	19.8	13.463
18,000-18,700	94.5	17.09	0.10	700	24,500	32,181	450	71.4	35	2.04	7.9	8.734
0-18,700	94.5	17.09	15.01	18,700	654,500	32,181	450	71.4	35	2.04	44.0	30.693
Ward S. connection, Dec. 13, 1908	87.8	11.147				South lateral.						
50 feet S. of H. & N. fan	108.86	14.184	3.037	2,720	89,760	10,143	161	68	33	1.90	20.59	14.863
North lateral partition	79.2	4.948				North lateral.						
C. & C. shaft connection	93.2	15.429	10.481	921	30,393	3,378	54	63	33	1.90	69.89	48.755
Drifts.												
2,350 west drift, section K	84.2	11.00										
2,350 west drift, near winze	98.6	18.77	7.765	530	12,190	11,667	413	28.25	23	1.23	561.4	391.62
2,350 east drift, end of pipe	89.6	5.93										
2,350 east drift, turn of drift, C.C.	97.3	7.74	1.806	165	3,877	1,996	63	31.5	23.5	1.34	55.8	38.923
2,150 N. drift, section A	87.4	4.68										
No. 6 fan	98.6	7.123	1.443	660	15,510	7,024	223	31.5	23.5	1.34	39.2	27.345
Shafts.												
C. & C. surface	38.5	0.868										
C. & C., 2,150	59.7	3.944	3.076	1,900	82,650	33,212	481	77	43.5	1.77	74.17	51.739
Yellow Jacket, surface	34.7	1.036										
Yellow Jacket, 900 level	49.6	3.760	2.73	900	33,750	10,665	166	64.2	37.5	1.71	51.76	35.106
Union S., surface	46.8	0.766										
Union S., 2,000 level	65.1	5.688	4.922	2,000	134,000	13,814	109.6	126	67	1.88	30.4	21.206

the surface and 109.2 underground. With the Ophir fan in use the total horse-power approximates 263.2.

11. *Air-Pipes.*—The extensive ues of air-pipes warrants a detailed description.

The air-pipe is in 12-ft. lengths, 11-in. and 15-in. diameter being commonly used. No. 20 galvanized iron, in sheets 36 in. wide, cut to the necessary lengths for the diameters required, and with longitudinal seams punched, is delivered to the mine shop and there made into pipe as required. Longitudinal seams are riveted with $\frac{3}{16}$-in. rivets, 2.5-in. pitch; girth seams, 4-in. pitch. Longitudinal and girth seams are soldered. Joints are made with a bell, which is riveted and soldered to the pipe. Laps are 0.75 to 1 in. The mouth of the bell is strengthened with $\frac{3}{16}$-in. wire. The diameter of the mouth is 16 in. and the depth 4 in., not including the lap. This gives an over-all length of 12 ft. for each section.

The bell-joint is known as the "Southwell" joint. Elbows are made with an 18-in. radius, measured to the center of the pipe, and are provided with a bell-joint; the inside diameter of the mouth is 15.5 in.; the outside, 16.25 in. Rivets on transverse joints are pitched 2.5 in., and on longitudinal joints 1.5 in. Six pieces, including the bell, are required for each joint. The opposite end of the elbow has a diameter of 15.75 in. Right-angle tees, used for branches, are provided with two 11-in. dampers, having a clearance of $\frac{3}{16}$ in., and 15-in. dampers with a clearance of from $\frac{1}{8}$ to $\frac{1}{16}$ in. The dampers are riveted to a flattened $\frac{5}{8}$-in. iron rod, one end of which is bent so as to form a 6-in. handle. The damper is cut in the blacksmith-shop from $\frac{1}{8}$-in. sheet iron. Elbows of 45° are made from three pieces, including the bell. "Y" pieces and special breechings for the fans are made as required; 20-in. pipes, occasionally used, are made from No. 18 or No. 20 galvanized sheet-iron, with longitudinal seams riveted on 2.5-in. pitch, and girth seams from 5 to 6.75-in. pitch. The inside diameter of the bell-mouth is 20.75 in.; the outside, 21.25 in.; and the depth of the bell is 5 inches.

Air-pipes are suspended by U-shaped iron straps, made from $\frac{3}{16}$- and $\frac{1}{4}$-in. round iron, each end of the U being provided with a 3-in. point. The straps are driven into the timbers, three straps to each 12-ft. section. Where the air-pipe is placed

in untimbered drifts, it is suspended by ropes or wire from plugs driven into drilled holes, or else from sprags wedged into place, and into which the iron strap-points are driven.

The bell-joints are made tight by wrapping several times with a tarred canvas strip 7 in. wide. Six or seven wrappings of 0.25-in. tarred cord are used to hold the canvas in place. New pipe, carefully wrapped in the manner described, allows very little air to leak through at the joints. Data on leakage will be given later in this paper.

If protected from acid mine-waters, and not battered by blasting, the life of an air-pipe is indefinite; but in wet shafts or where exposed to acid waters they frequently have to be replaced.

The shop-cost of 15-in. air-pipe is $8.50 per section; the cost of installation is nominal. The placing of a 15-in. air-pipe approximates $9.25 per section, or $0.77 per linear foot.

Air-Pipe Leakage.—Leakage depends very largely upon the care taken in wrapping the joints. If the ends are much battered the leakage may be considerable. Pressure and velocity of the air are minor factors.

A well-wrapped air-pipe extends from the Mint shaft to the Hale and Norcross workings. Connection is made with a fan at a distance of 1,500 ft. from the shaft. The discharge from this fan, at a distance of 230 ft., showed a loss of 411 cu. ft. per min., or a delivery of 89 per cent. of the inflowing air. The loss per joint is 2.93 cu. ft. per min. A water-gauge at the Mint shaft showed a pressure into the air-pipe of $\frac{13}{16}$ in. The continuation of the pipe, 267 ft. of 15-in. and 268 ft. of 11-in., or 535 ft. in all, gave an additional leakage of 1,941 cu. ft. per min. The pipe was not well wrapped, and the loss per section figures out 44.1 cu. ft. per min. In this case the suction created by the fan undoubtedly reduced the leakage on the 1,500-ft. section.

In the north lateral of the Sutro tunnel 120 ft. of 20-in. air-pipe showed a leakage of 122 cu. ft. per min., or 12.2 cu. ft. per section. The pipe was not wrapped, but was very carefully placed together.

In the Gould and Curry mine a leakage in 825 ft. of 15-in. and 11-in. pipe measured 437 cu. ft. per min., or 6.4 cu. ft. per min. per section. The pipe was partly wrapped and the air-pressure measured $\frac{13}{16}$ in. The fan was not in use.

In the Savage workings, with fan in operation and 15-in air-pipe, the leakage amounted to 1,610 cu. ft. per min. The length of 555 ft. of 15-in. pipe beyond the fan and 180 ft. of 11-in. pipe gives a leakage of 26.4 cu. ft. per min. per sectión. In this case most of the leakage took place on the last 180 ft. of 11-in. pipe.

Dampers showed a variable leakage, depending upon the perfection of the fitting and the care used in closing. Measurements showed from 10 to 180 cu. ft. per minute.

Two canvas curtains were measured: one showed a leakage of 1,635 cu. ft. per min. over an area of 35 sq. ft.; the other, 3,980 cu. ft. per min. over an area of 20 sq. feet.

The general conclusion is that leakage can be controlled within narrow limits by carefully wrapping the joints. A mile of 15-in. pipe with a leakage of 2.93 cu. ft. per min. per section would give a total leakage of 1,289 cu. ft. per min., and with an inflow of 4,000 cu. ft. per min. a delivery of 67 per cent. of the air is possible. This would be more than sufficient to ventilate an ordinary working-face. It is not always desirable to have a pipe tightly wrapped up to the face. Mr. Higginson informed me that with pipe tightly wrapped the blasts, if heavy, will cause much loss by collapsing. The practice is to wrap the pipe up to within from 200 to 300 ft. of the face and then leave the remainder unwrapped. Old and battered pipe is used close to the face.

Stops are constructed from timber or canvas; 1-in. rough boards, battened, are commonly used; doors are made from the same material; clay is used to stop small cracks and openings.

V. Measurement of Ventilating-Currents.

1. *Temperature.*—The mine-measurements were made with an ordinary Fahrenheit thermometer of good grade. The instrument was suspended from a timber, and in approximately the center of the air-way. Unfortunately, this thermometer was not available for calibration. My observations were made with chemical thermometers carefully matched and compared.

2. *Velocity.*—All upcast shafts are housed in by wooden chimneys, which conduct the air outside the shaft-houses. Access to a shaft is had through doors, and from them most of the section may be reached. Five measurements were made in

as many positions in each compartment. The center and four corners were the positions selected, as shown by the numbered circles in Fig. 10. The corner position was taken 1 ft. from the timbers each way. The anemometer was held stationary in each position, and the velocity determined over a time-interval varying from 0.5 to 2 min. Observed velocities were corrected by the calibration-curve furnished by the maker of the instrument. Measurements were not taken by moving the instrument over the whole section on account of the difficulty of reaching some of the compartments. In part of the time during which measurements were made no work was done in the shafts, but during the remainder shaft-crews were at work in all three upcasts. The housing of the shaft at the collar prevented eddy-currents, and consequently all measurements were made in the plane of the shaft-collar. The time required to make from 15 to 20 separate measurements on one shaft varied from 30 to 50 min. Table XVIII. presents a complete set of observations for the Combination shaft.

The down-cast air was measured in one shaft only and at one time, since in all of the down-cast shafts work was going on. Difficulty was experienced on account of eddy-currents, even though the anemometer was held some distance below the collar of the shaft.

The center velocity of each compartment generally exceeded the corner velocities. The ratio between the two was calculated and an average of 1.19 obtained for the Combination, 1.09 for the Ophir, and 1.19 for the Belcher shaft. The reciprocal of 1.19 is 0.84, and this was used to calculate average velocities from center velocities. As an example of the use of this method for determining the average velocity, the following figures are given: The Ophir shaft was measured by both methods, Dec. 27, 1908. The average of five readings for each compartment was 653, 652 and 675 ft. per min., and the center velocities were 778, 732 and 732. Computing from these figures by the use of the constant 0.84, the following results were obtained: 690, 651 and 651 ft. per min. The average velocity by the 15 readings was 660, and the calculated velocity 664 ft. per minute.

Sections were taken in straight portions of the drifts where-

ever these were available, and timbered sections were used in preference to the untimbered, but no special steps were taken to prepare sections of uniform cross-section. The anemometer was slowly moved over the whole section for a period varying from 0.5 to 2 min. Duplicate readings were generally taken.

On the whole, the measurement of the air in the air-ways of a mine, without special provision for smooth sections, is approximate only, and it is not possible to make very accurate measurements without resorting to extreme measures. My measurements were made during the working-hours of the mines, and without special provision for smooth sections.

The anemometers were of the Biram type, one 4-in. reading to 1,000 ft., one 4-in. reading to 100,000 ft., and one 6-in. reading to 1,000 ft. The recording-anemometer was a Julien P. Friez instrument, of the type used at the government meteorological stations. A sling-psychrometer was used in determining humidities, and the psychrometric tables published by the U. S. Weather Bureau were used in reducing results.

VI. Cost and Operating-Expense.

It is almost impossible to obtain detailed figures on the cost of the ventilating-system, but fairly accurate estimates may be made. The following estimate was made by taking the cost of motors and fans and adding 20 per cent. for cost of installation. The 15-in. air-pipe in place was estimated as $0.77 per ft. and 11- and 20-in. proportionally:

Cost of supplementary ventilating-plant:	
Motors,	$6,000
Fans,	6,000
Air-pipe,	10,000
	$22,000

Note.—The above does not include the cost of the fan and pipe in the Overman and Caledonia mines, nor the cost of electrical conduits.

Depreciation, maintenance, and operating-expense:	
Depreciation and maintenance assumed as 10 per cent. of plant-cost, or $2,200 per annum, or per month,	$183.30
Operating-cost:	
Power: 165 h.p., at $5 per h.p. month,	825.00
1 electrician, at $4 per day,	120.00
Incidentals,	50.00
Total expense per month,	$1,178.30

Note.—Shift-bosses and miners take immediate charge of fans and motors, and no extra men are employed except the electrician.

The expense of maintaining the upcast shafts may be properly charged against the cost of ventilation, and this is a large item, but at present no figures are available. During the past year shaft-crews of from four to six men have been employed on single shafts in all three upcast shafts for periods of from two to six months. On an average this work has to be done every second year.

On the foregoing estimate of expenses per month, the cost of each 10,000 cu. ft. of air passing through the mines is $0.00116, not including the cost of repair of upcast shafts.

VII. Underground Humidity.

In summer the air entering during the day is of low and in winter of moderate humidity. The temperature is neither excessively low nor high. The greater heat of summer is partly compensated for by the lower humidity, while the higher humidity of the winter months is compensated for by the lower temperature. The temperature of the entering air is of more importance than the humidity. The air-current begins to rise in temperature and absorb moisture as soon as it enters the underground workings. How rapidly it changes depends upon the velocity of the air-current, the difference of temperature between the air-current and the surrounding rocks, the relative humidity, and exposure of the air-current to moisture. The effect of these factors may best be studied from the three following examples:

1. *The Sutro Tunnel.*—Table II. and Fig. 2 are referred to. The temperature of the incast air reaches a maximum at a point 11,000 ft. from the portal. The temperature of the water controls the air-temperature. Humidity begins to increase as soon as the air enters the tunnel. For the first 6,800 ft. the air does not come in contact with the drainage-water, and consequently the moisture must come mainly from the walls.

The amount of moisture from this source is 43.8 grains per square foot of exposed surface per hour. Much of this part of the tunnel is untimbered. At the 6,800-ft. station the air comes in contact with the drainage-water, and absorbs moisture until it is almost saturated at a point 10,000 ft. from the portal, or after moving over the water for a distance of only 3,200 ft. The approximate exposed water-surface is 19,200 sq. ft. (average width

of stream taken as 6 ft.). On the basis of the exposed water-surface the rate of accession of water-vapor is 798.5 grains per square foot per hour. The rapidity with which the air takes up moisture is remarkable. The rates for different parts of the tunnel have been calculated and are to be found in Table XXV. The total exposed surface, both rock walls and water, has been taken in calculating the table in preference to the water-surface

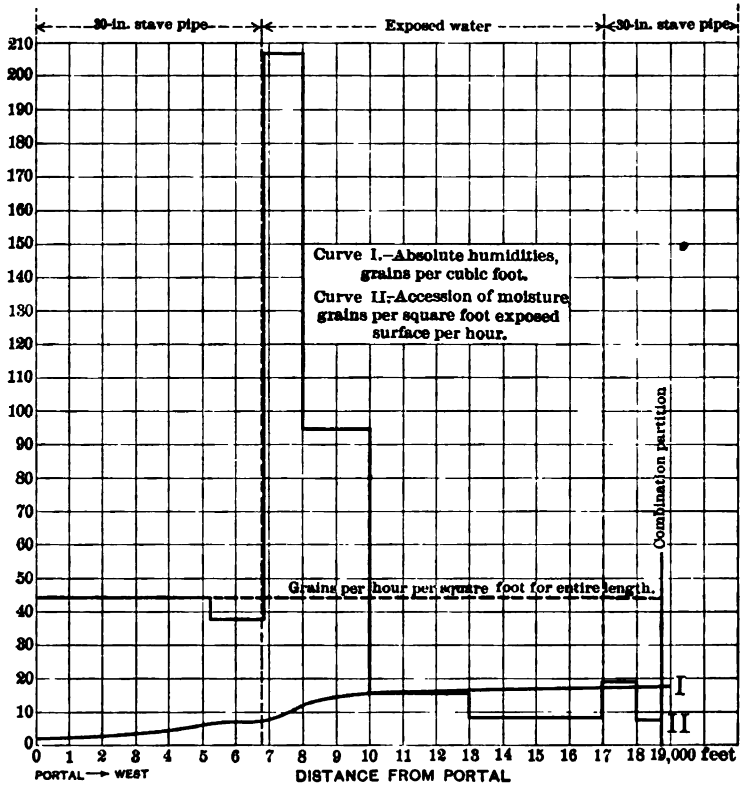

FIG. 14.—CURVES OF ABSOLUTE HUMIDITIES AND RATE OF ACCESSION OF WATER PER HOUR IN THE SUTRO TUNNEL, Nov. 22, 1908.

alone. The results have been also graphically represented in Fig. 14. The rate for the whole tunnel up to the Combination partition is 44 grains per square foot per hour. For purposes of comparison, both north and south laterals for a portion of their length were calculated. The south lateral is timbered for about one-half the distance taken. Very little water drops from the walls, and the drainage-water is carried in a stave pipe, so that

practically all the moisture comes from the walls. A rate of 20.59 grains was found. The north lateral, on the other hand, has open drain-boxes and more or less water is present, although the main flow of drainage-water is carried in stave pipes. The rate is 69.89 grains.

2. *Mine-Drifts.*—Three drifts have been calculated: the very hot west drift of the 2,350-ft. level; the east drift of the same level; and a portion of the 2,150-ft. drift. Although the west drift of the 2,350-ft. level is snow-shedded, the hot vapors find their way through with considerable freedom and give a high rate, 561.4 grains per square foot per hour. The east drift of this level is comparatively dry and practically out of the hot zone. It gives a much lower rate, 55.8 grains. The 2,150-ft. drift is also snow-shedded, but, since the lower workings have drained off the hot water to a great extent, the accession of moisture is very much less, 39.2 grains per hour.

3. *Down-Cast Shafts.*—Three down-cast shafts have been calculated: the C. & C. down to the 2,150-ft. level; the Yellow Jacket to the 900-ft. level; and the Union to the 2,000-ft. level. The range of temperature and the relative humidity at different levels in these shafts are shown in Fig. 15.

The C. & C. shaft is not a very wet shaft, but considerable moisture from the drainage-water is present from the 1,750-ft. to the 2,150-ft. level, which accounts for the high rate of 74.17 grains per square foot per hour. The Yellow Jacket is moderately wet, while the Union is quite dry. The rates are 51.76 and 30.4 grains respectively.

A better idea of the amount of water taken up by an air-current under high-temperature conditions may be gained by considering the total amount of moisture discharged with the air at the upcasts. The total amount in all upcasts is 519,788 lb. per 24 hr., divided as follows: Combination, 146,330; Ophir, 172,140; Belcher, 58,941; Ward, 142,377 lb. These quantities have been calculated on the average amounts of air as given in Table III. Assuming an average temperature of 60° F. and a relative humidity of 50 per cent., the down-cast air brings into the underground workings 127,990 lb. of moisture per 24 hr., which leavea 391,798 lb. of water per 24 hr. as the amount taken up by the air-currents. The calculated result is somewhat less than the actual amount.

The unit used in comparing different mine-workings is the rate of accession of water vapor per square foot of exposed surface per hour, and is calculated from the following ratio:

$$\frac{\text{Volume of air in cubic feet per minute} \times \text{increase in absolute humidity per cubic foot for the length under consideration} \times 60\text{ min.}}{\text{Square feet of exposed surface in length under consideration, or length of air-way} \times \text{perimeter.}}$$

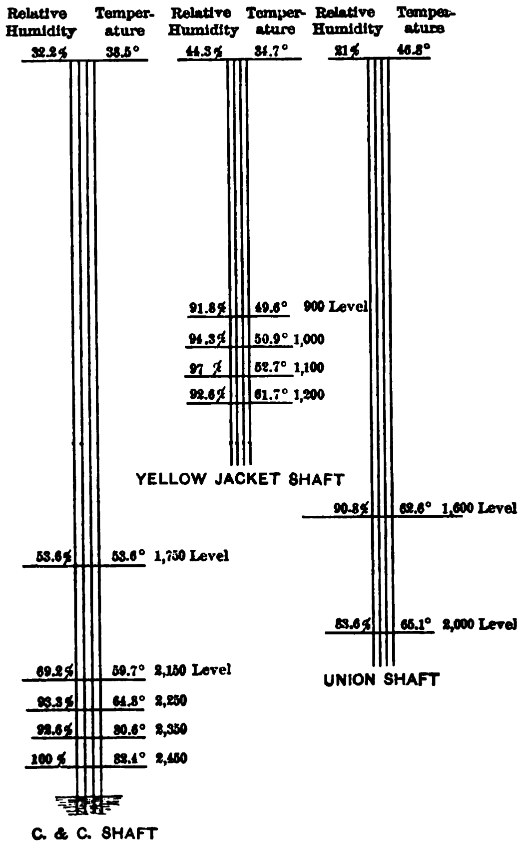

Fig. 15.—Range of Temperature and Relative Humidity in Down-Cast Shafts.

VIII. Control of Humidity and Heat.

Control of humidity and heat is a matter of considerable importance as affecting the efficiency of the miners. High temperatures and saturated air are conducive to low labor-efficiency, while, on the other hand, high temperature with low humidity does not lower efficiency to an unreasonable extent. The ex-

amples in the previous section show that if water can be kept away from the air a moderate humidity is possible, but this cannot always be done. Careful boxing of drains and boarding-off of hot water are beneficial; the carrying of large quantities of drainage-water in closed conduits, such as the stave pipe, is very effective, but, on the whole, the mine-drift or tunnel cannot be considered an ideal air-way under high-temperature conditions. However carefully the drainage-water may be carried, the emanation of water-vapor from the walls and surfaces of the timbers still remains as an important humidifying-agent. The construction of moisture-tight walls for the drifts is out of the question, and the use of excessive volumes of air is impracticable. The carrying of air through sheet-metal conduits of moderate cross-section is the only practicable way of delivering air-currents of low humidity to working-places where a high temperature prevails, and this method is in general use upon the Comstock. The small size of the air-pipe as compared with the drift requires the use of power-driven fans to force a sufficient quantity of air. The working-face of a dead end, such as a drift, is perhaps the most difficult to maintain in proper condition, and as a consequence a number of measurements were in the ends of drifts ventilated by pipes. These measurements are given in Table XXVI.

With the exception of two cases the air is delivered practically either at the temperature of the working-face or a little above. The two exceptions indicate that if a sufficient volume is forced through the pipe the air can be delivered lower in temperature than the working-face, although usually the volume is such that even with short distances the air is delivered at the same temperature as the drift through which the return current is passing. The quantity of air varies from 500 to 1,200 cu. ft. per min., in ordinary cases, up to from 2,000 to 4,000 cu. ft. per min. in cases in which an excessive temperature is to be overcome. The humidity of the discharged air depends upon the humidity of the air entering the pipe or fan and the temperature of the discharged air, and for this reason the air should be drawn from a current low in both temperature and humidity. The relative humidity of the discharged air ranged from 20 to 41 per cent., while the relative humidity at the face, usually 20 to 25 ft. beyond, ranged from 22 to 74.4 per cent.

TABLE XXVI.—*Air- and Temperature-Measurements at Drift-Ends.*

Place.	Temperature of Discharged Air.	Temperature of Working-Face.	Relative Humidity of Discharged Air.	Relative Humidity of Working-Place.	Distance of Working-Face from Pipe-End	Cubic Feet of Air per Minute.	Velocity of Discharge.	Diameter of Air-Pipe.
	Deg. F.	Deg. F.	Per Cent.	Per Cent.	Ft.		Ft. per Min.	In.
Large Volumes of Air.								
2,350 C. & C. shaft, west drift	82.4	110	....	100	25	4,160	3,492	15
2,350 C. & C. shaft, east drift	89.6	96.8	40.6	50.7	40	1,996	2,050	15
Moderate Volumes of Air.								
Gould and Curry, Sutro tunnel	91.0	91	31	31	12	989	1,521	11
Savage, 75 ft. above Sutro tunnel	92.6	92.1	37.3	51	30	894	1,356	11
Chollar, Sutro tunnel	104.5	....	23.8	....	....	934	1,416	11
Chollar, Sutro tunnel	106 2	106.2	22	22	10	721	1,104	11
Hale and Norcross, cross-cut	106.1	107.6	21.8	32.2	15	941	1,426	11
Hale and Norcross, cross-cut	109.4	109.6	20.0	29.8	25	688	1,054	11
2,000 level Union, NW. cross-cut	99.1	98.6	38.3	43.4	20	1,271	1,042	15
2,000 level Union, south drift	98.6	....	40.8	....	30	971	796	15
2,350 C. & C.. north workings	104.5	....	36.5	....	....	544	490	15
1,200 level, Caledonia	86.0	87.8	41.0	74.4	25	792	1,200	11
No Active Circulation of Air.								
2,000 level, Union, east cross-cut	....	100.9	....	74.8				
1,100 level, Yellow Jacket, north drift to Imperial	....	89.9	....	86				

The first drift in the table, the west drift of the 2,350-ft. level of the C. & C., illustrates abnormal conditions. Great heat and a large quantity of moisture emanated from the rock walls, which had not at the time been closed off by boarding, and this charged the air-current to saturation and raised the temperature 27.6° F. in a very short distance. The east cross-cut of the Union on the 2,000-ft. level is a dead end, in which practically no air was circulating, and which was also dry. The north drift on the 1,100-ft. level of the Yellow Jacket is also a dead end, without circulating air, but with a small amount of water on the floor of the drift. In neither of these drifts was the air saturated with moisture, but in both it was quite oppressive.

The principle and use of the air-pipe as a means of delivering low-humidity air-currents is of course no new thing. Church comments upon the efficiency of the system, and in particular notes that by forcing the air through metal pipes the relative humidity of the air-current is lowered and it becomes quite effective in absorbing the excessive perspiration of the miners. The only improvements to be noted in Comstock practice since the time of Church's observations, about 30 years ago, are an increased velocity and a larger quantity of air, larger air-pipes, the use of high-speed metal fans instead of the slower wooden fan, the use of electric motors instead of compressed-air engines, and the use of more power.

Supersaturation of air-currents, resulting in the production of fog, is encountered in all of the upcast shafts near the surface, and in all places underground where cool currents mix with hot saturated air. The moisture condenses on timbers and rock-surfaces, and frequently causes the rock to swell, disintegrate, and cave. By stimulating the growth of fungi it also indirectly causes the timbers to rot. In only one place is any attempt made to remedy such a condition, and in this case a small amount of cool air is introduced into the air passing up the Ophir incline. Trouble was experienced in this incline by the condensed moisture causing the "back" to swell, and the remedy was found effective. It is evident that the introduction of cool air lowers the temperature of the upcast air and thus reduces the efficiency of the upcast, and if too much air is introduced the opposite from the desired effect would be obtained. Passing the air through the shortest and driest

air-ways to the upcast, and the careful stopping of all leaks of cool air, would be a more effective way than that now followed.

Some observations upon the cooling-effect of large volumes of air were made. The most striking results were in the case of the hot west drift on the 2,350-ft. level of the C. & C. Before the connection was made the maximum temperature of this drift varied from 126° to 130° F., and this in spite of forcing 4,160 cu. ft. of air per minute into the face. The air in the drift was saturated and would not support a candle-flame. After making the connection and before boarding-off the walls of the drift, the temperature dropped to 110°, with a volume of air passing of 10,350 cu. ft. per min. The air-current was supersaturated. After snow-shedding, the temperature dropped to 98.6°, with 11,667 cu. ft. per min. passing and a relative humidity of 98.8 per cent. In the Hale and Norcross cross-cut, on Dec. 28, the rock-temperature measured 113° and the air-temperature 109.6°. The reduction in temperature of more than 3.4° was effected by an air-current of 688 cu. ft. per minute.

The most effective means for the control of an excessive temperature is the tight boarding of water-boxes carrying hot water, the use of boards and battens for the sides and tops of drifts, and passing large volumes of air as cool as possible.

IX. Physiological Effect of Working in High Temperature and Humidity.

Much has been written regarding the effects upon the human system of poisonous and other gases, but very little is to be found in mining-literature concerning high temperature and humidity. Le Neve Foster says:[5]

"In still and saturated air it is hardly possible for men to do hard continuous work above 80° or 85° F., even when stripped to the waist. At higher temperatures in saturated air the amount of work possible becomes less and less, and the body temperature may rise rapidly, though men accustomed to the heat can bear it much better than others. At temperatures above about 90° by the wet bulb it is only possible to work for short periods, and it becomes difficult even to remain without working. Thus, at a temperature of 93° in still and saturated air, I found that, though I was stripped to the waist and doing practically no work, my temperature rose 5° in two hours, and was still rising rapidly when I found it necessary to come out. On the other hand, it is a well-known fact that if the air is dry much higher temperatures can be borne with ease and comfort. In collieries

[5] *Investigation of Mine Air*, Sir C. Foster and J. S. Haldane, p. 151 (1905).

where the air is fairly dry and in motion, men can work well at a dry-bulb temperature of 90°, or even 100°, and in hot climates with very dry air much higher temperatures are not oppressive."

Eliot Lord[6] discusses the permanent effect of high heat upon the system, and reaches the following conclusion:

"The ultimate effect of this extreme heat on the miners' constitution is not so easily noted. The mine levels differ so materially in temperature, and the assigned station of a miner is so frequently changed from one cause and another, that it is impossible to obtain at present complete comparative data. That prolonged labor in a hot, impure atmosphere will assuredly shorten life appears indisputable; but whether the system is permanently or materially injured by intermittent working under those conditions is more questionable. The power of recuperation appears extraordinary, and, unless the strain is intense and frequent, no lasting injury may be inflicted. The limits of permissible strain will, of course, vary with the relative power of endurance. The action of all the bodily organs appears to be stimulated by the heat, with the exception of the stomach alone."

Messrs. Haldane and Thomas[7] comment on exposure to high temperatures and sudden variations of temperature as follows:

"Miners are commonly exposed to high temperature underground, and to comparatively sudden cold on coming out, often with damp clothes on. Much stress has been laid on this fact, particularly in relation to lung diseases. Nevertheless, this cannot be an important cause of lung disease, for colliers are similarly exposed. Moreover, in England, colliers never wash and change their clothes at the pit-head, while Cornish miners almost invariably do so, in a heated building ('dry') provided on the mine. The effects of high underground temperatures on men and horses are certainly of considerable interest and economic importance, and we know from personal observations that in warm and moist air underground the body temperature often rises several degrees; but we can find nothing in the circumstances connected with the mode of occurrence of miners' phthisis to suggest that high underground temperatures are in any way connected with its causation."

Church writes[8] on the immediate results of high temperatures and hot vapors, as absent-mindedness, dizziness, fainting, vomiting, and, as graver results, insanity and death. His final conclusion is the following:

"The casualties positively traceable to the heat are therefore twelve per cent. of the whole. Probably the heat increases the bad effects of powder fumes and natu-

[6] Comstock Mining and Miners, *Monograph IV.*, *U. S. Geological Survey*, p. 400 (1883).

[7] *Transactions of the Institution of Mining and Metallurgy*, vol. xiii., p. 383 (1903–04).

[8] Accidents in the Comstock Mines and Their Relation to Deep Mining, *Trans.*, viii., 95 (1879–80).

ral gases, and by making repairs to the shafts more frequently necessary it indirectly adds to the occasions when disasters may occur. I also confess to the belief, which is not sustained by observations upon specific casualties, that some allowance should be made for a less active mental condition, a dulling of the faculties, and a certain recklessness to which the heat sometimes goads the men. On the other hand the heat makes them more cautious except when under momentary impulses, and I have never seen American miners more careful of themselves than in these mines. On the whole the good and bad effects of the heat seem to nearly balance each other, and I think that an allowance of five per cent. for the casualties indirectly caused by the high heat would be sufficient."

The main facts brought out by these observers are that high temperatures and humidities cause, under some conditions, a notable rise in body-temperature; that all the bodily organs with the exception of the stomach are stimulated; that prolonged exposure to such conditions undoubtedly lowers vitality; that intermittent exposure produces no permanent ill-effects; that these conditions cannot be considered an important cause of lung-diseases; and, lastly, that under abnormal conditions loss of mental control and serious bodily disturbances result.

Local physicians inform me that the average life of the Comstock miner approximates 25 years, and that the miners do not show any greater susceptibility to any particular disease than the residents of the town. I am acquainted with miners who have worked more or less continuously underground for 30 years, and who are still capable of doing a good day's work. Compared with miners of other districts the Virginia City miners may be said to be just as healthful, if not more so. The result is due in a large measure to the fact that the Virginia City miner observes certain precautions, and that the mine-managements provide the necessary facilities. Miners are careful not to expose themselves to cold drafts, since they work stripped to the waist. On passing from a hot to a cold place a heavy coat is used to protect the heated body. Wet clothes are either removed before going to the surface, or trousers and coat worn over them. All miners take hot and cold showers after coming off shift. Frequent drinking, and the bathing of the hands, wrists, arms, and head in ice-water are resorted to in all hot workings, and undoubtedly serve to keep down the body-temperature, while temporarily refreshing. Frequent rests are taken in special cooling-rooms, so placed in the workings as to receive the freshest and coolest air. In exceedingly hot work-

ings cold water from a hose is turned upon the miner while at work.

The effect of the underground conditions upon the blood was made the subject of a preliminary study by Prof. P. Frandsen at my request. His results are of importance, and in consequence are given in full in the accompanying note. His conclusion, given tentatively, is that "the conditions in the deep workings of the Virginia City mines do not have any particularly detrimental effects upon the composition of the blood. From a comparison of Tables XXVII. and XXIX. it appears that the main permanent effect is an increased hæmoglobin-content of the individual red blood-corpuscles."

Conditions are not favorable for a study of the comparative efficiency of miners working under normal temperatures and under those in the lower levels of the Comstock. The following conclusions are the results of my underground experience: Moderately high temperatures, from 95° to 105° F., with moderate humidities, from 50 to 70 per cent. relative humidity, and with air-currents of velocities from 200 to 300 ft. per min., do not prevent efficient work nor are they particularly uncomfortable.

A higher temperature, from 110° to 115°, together with the same conditions as above, decreases efficiency to a considerable extent.

A high temperature, from 110° to 115°, with high humidity and moderate velocity air-currents, very greatly impairs the miners' efficiency; and a still higher air-velocity, under the same conditions, renders workings more bearable, but miners cannot work very long at one time.

A moderately high temperature, from 95° to 105°, in a saturated atmosphere with no current, becomes very trying. Prolonged exposure with much exertion is dangerous.

A moderate temperature, from 90° to 98°, and saturated air-currents of a velocity of from 400 to 500 ft. per min., with more or less vitiated air, are conditions which are very trying and give a low labor-efficiency. Vitiated air will impair labor-efficiency to a greater extent than a high temperature.

X. APPENDIX.

By PROF. P. FRANDSEN.*

With a view to obtaining more exact information as to the physiological effects of the atmospheric conditions in the lower levels of the Virginia City mines, some preliminary studies were made upon the blood of subject No. 1 and several miners. To determine the effect of a temporary sojourn in the mines, blood of subject No. 1 was examined at Reno, December 10, two days before going to Virginia City; December 12, after remaining 2.5 hr. in the 2,000-ft. level of the Union shaft; on December 13, after spending 4.5 hr. in the Sutro tunnel and south lateral; and on December 14, at Reno. The temperatures in the workings visited ranged from 100° to 109°, with humidities of from 60 to 100 per cent. During both trips the subject and myself perspired profusely while underground, but felt no ill-effects other than a slight dizziness and a natural fatigue. The results of the examinations are given in Table XXVII.

TABLE XXVII.—*Subject No.* 1.

Date.	Place.	Red Corpuscles per c.c.	Leucocytes (white corp.) per c.c.	Hæmoglobin.	Hæmoglobin Index.
				Per Cent.	
Dec. 10...	Reno........................	5,160,000	7,111	95	92
Dec. 12...	Va. City, underground	4,820,000	12,333	...	...
Dec. 13...	Va. City, underground	6,496,000	12,277	100	77
Dec. 14...	Reno........................	6,220,000	9,888	95	76

The difference in red-counts on December 10 and December 12 may be ignored as too slight to have any significance. On December 13, however, after the longer sojourn underground, there was an appreciable increase of red corpuscles, which is still to be noted on the following day. On both occasions there was a considerable leucocytosis, due mainly, as Table XXVIII. shows, to an increase in the number of neutrophiles.

Tables XXIX. and XXX. give the results of examinations of five miners who have been engaged in mining for 10 years or more, and for a year or more have been at work under the conditions described in this paper as characteristic of the deeper

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TABLE XXVIII.—*Differential Count of Leucocytes (White Blood-Corpuscles).*

Date.	Leucocytes per c.c.	Percentage of Varieties.				
		Lympho-cytes.	Mononu-clears.	Neutro-philes.	Eosino-philes.	Mast.
Dec. 10..	7,111	41.5	2.5	55.5	0.4	0.1
Dec. 12..	12,333	32.0	2.0	65.0	0.9	0.1
Dec. 13..	12,277	32.8	2.4	64.1	0.5	0.2
Dec. 14..	9,888	44.6	3.7	49.0	2.5	0.2

levels. All the subjects were in a fair state of health, and men of good physique, well nourished, and moderate users of tobacco and liquor. The examinations were made as soon as the men reached the surface at the close of their day's shift, and, except in the case of subject No. 2, before the shower which all take at the end of the day's work.

TABLE XXIX.—*Summary of Examinations of Blood of Miners.*

Subject.	Age.	Red Corpuscles per c.c.	Leucocytes per c.c.	Hæmoglobin.	Hæmoglobin Index.	Pulse.	Temperature.
				Per Cent.			
2	38	4,236,000	6,000	115	1.35	80	97.4
3	29	5,386,000	7,814	120	1.11	76	97.2
4	33	4,460,000	10,777	120	1.34	70	98.2
5	38	5,178,000	7,055	111	1.07	80	98.2
6	38	5,422,000	12,277	114.5	1.05	61	99.1

TABLE XXX.—*Differential Count of Leucocytes of the Blood of Miners.*

Subject.	Leucocytes.	Percentage of Varieties.				
		Lympho-cytes.	Mononu-clears.	Neutro-philes.	Eosino-philes.	Mast.
2	6,000	36.8	5.8	56.6	0.4	0.4
3	7,814	52.4	3.2	48.5	0.5	0.4
4	10,777	40.7	3.5	54.0	1.0	0.8
5	7,055	53.3	2.3	43.1	0.8	0.5
6	12,277	34.3	3.7	59.1	2.8	0.7

The most striking feature revealed by Table XXIX. is the uniformly high hæmoglobin index of the miners' blood. The

percentage of hæmoglobin was determined by means of Dare's instrument. In two cases only is the number of red corpuscles below the normal average, 5,000,000 for persons at sea-level; but the reduction is hardly great enough to be considered indicative of anæmia. Moreover, microscopic examination of stained preparations showed no abnormal appearances. The red corpuscles were of uniform size and shape, and chiefly noteworthy for their richness in hæmoglobin, indicated by the intensity and uniformity with which they took the eosin stain. The foreman said of subject No. 2 that he did not stand the heat as well as the other men, and had fainted on several occasions.

Except possibly in the cases of subjects Nos. 1 and 6, the leucocyte-counts are within the limits of variability in normal individuals. Most striking is the high percentage of lymphocytes, particularly in subjects Nos. 3 and 5.

As living in high altitudes is claimed by many authorities to bring about a marked increase in the number of red corpuscles over the number normal to residents of the sea-coast, I give in Table XXXI. for comparison with Tables XXVI. to XXX. the results of counts made upon subjects resident in Reno and Virginia City, all of whom were in good health and had resided in the locality designated for a number of years.

TABLE XXXI.—*Summary of Blood-Examinations in Relation to Altitude.*

Subject.	Age.	Red Corpuscles per c.c.	Leucocytes per c.c.	Hæmoglobin.	Hæmoglobin Index.	Place.	Altitude.
				Per Cent.			Ft.
1	32	5,160,000	7,111	95	92	Reno.	4,553
7	32	5,399,000	8,944	105	97	Reno.	4,553
8	33	5,536,000	8,944	109	98	Va. City.	6,000
9	40	6,074,000	6,777	104	85	Reno.	4,553
10	20	6,378,000	4,111	110	86	Reno.	4,553

Subject No. 9 for several years has been making regular and frequent trips to the summit of Mount Rose, elevation 10,800 ft., and is an experienced mountain-climber. The examinations of both Nos. 9 and 10 were made about 24 hr. after their return from a fatiguing three days' trip to the summit of this mountain.

These counts do not show as great an increase in the number

of red corpuscles as the published accounts of other observers would lead us to expect in these altitudes. In comparison with these results, the number of red corpuscles obtained in the counts on the miners would not seem to be very far from the normal. Differential leucocyte-counts were made on the blood of these subjects, as given in Table XXXII.

TABLE XXXII.—*Differential Count of Leucocytes in Relation to Altitude.*

Subject.	Leucocytes per c.c.	Percentage of Varieties.				
		Lymphocytes.	Mononuclears.	Neutrophiles.	Eosinophiles.	Mast.
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
1	7,111	41.5	2.5	55.5	0.4	0.1
7	8,944	33.4	4.5	57.6	3.5	1.0
8	8,944	32.2	5.4	60.4	1.7	0.3
9	6,777	44.4	5.2	49.2	1.0	0.2
10	4,111	42.0	5.5	50.0	2.1	0.4

The only point to be noted is the relatively high percentage of lymphocytes.

While these studies are too few to warrant an extended discussion or the statement of any very definite conclusions, they are of interest as indicating that the atmospheric conditions in the deep workings of the Virginia City mines do not have any particularly detrimental effects upon the composition of the blood. From a comparison of Tables XXIX. and XXXI., it appears that the main permanent effect is an increased hæmoglobin-content of the individual red corpuscles.

ACKNOWLEDGMENTS.

In collecting the foregoing data I received many courtesies from the officials of the Ophir mine, the Sutro Tunnel Co., the Yellow Jacket Mining Co., the Caledonia, and Overman Mining Cos. I wish especially to acknowledge the assistance of Dwight T. Smith, of the Virginia Mining School; T. McCormick, superintendent of the Ophir mine; T. Sullivan, of the Ophir mine; B. O'Hara, foreman of the Sutro tunnel; and Leon M. Hall, of the Ward Shaft Pumping Association. To Professor Frandsen, for his contribution, I am particularly grateful.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Conservation of Coal in the United States.*

BY EDWARD W. PARKER, WASHINGTON, D. C.

(Spokane Meeting, September, 1909.)

If one is to place any credence at all in the reports published in the daily press, the subject of conservation has been a very lively topic of conversation during the past 60 days, and it does not appear that the temperature of the summer months has been in any way moderated by the discussion. It is a subject in which we are all vitally interested and to which, so far as our mineral resources are concerned, both the Institute and the Geological Survey have liberally contributed. The report of the National Conservation Commission, appointed by President Roosevelt, contains a series of papers on the conservation of mineral resources, all of which were prepared by members of the Geological Survey and were compiled largely from information previously collected by that Federal bureau in the performance of its regular duties. It is not the purpose of this paper to reiterate *in extenso* any of the material already published. The contributions of the Survey officials to the Commission's report have been published in a separate document as *Bulletin No.* 394. This document is for free distribution and may be obtained upon application to the Director of the Survey. In the preparation of this paper I desire merely to make a few suggestions regarding the possible necessity of some restraint upon or control of one branch of the mining industry with which I have been somewhat closely associated for the past 20 years—that of coal.

Most of the members of the Institute are cognizant of the suits brought by the government against the anthracite-operators in Pennsylvania, or the combination of interests commonly known as the "hard-coal trust." No defense of any illegal combination in restraint of trade is intended, but there are some facts which should not be lost sight of, and unfortu-

* By permission of the Director of the U. S. Geological Survey.

nately those whose opinions are based upon the "news" given to us by the daily press are likely to be governed by *ex parte* testimony. The present situation in the anthracite-region is one that has been developed through sheer necessity, if the conservation of the supply of anthracite and the prolongation of the life of the fields in the best interests of the people were to be attained in any other way than through government control, and government control did not seem to be materializing. I believe that even Doctor Raymond will subscribe to the statement that a good part of the history of anthracite-mining has been one of profligate waste in the mining, preparation, and use of that precious supply of fuel; and this has only been remedied, none too soon, and could, under the circumstances, only be remedied, by the close control and conservative management which have been brought about in recent years. And I might pause here to pay a merited tribute to such men as Doctor Raymond, Eckley B. Coxe, P. W. Sheafer, Franklin B. Gowen, William Griffith, and a few others through whose efforts many reforms which lessened of the waste of anthracite were effected. They were the pioneers in the battle for conservation, and a monument should be erected to them.

The securing by the Reading R. R. for its offspring, the Philadelphia & Reading Coal & Iron Co., of the great coal-reserves it owns to-day, was the beginning of a great movement which was foreseen by those in a position to see. The Reading company was temporarily bankrupted through its guarantee of the debt thus incurred, but the possession and control of those coal-lands are indirectly the most valuable assets of the railroad at the present time. More than this, however, in the ultimate economy of things, has been the preservation of thousands of acres of coal-lands from reckless spoliation. The way was paved for the safe and sane control of the anthracite industry, albeit by a trust, and a stop was put to the cut-throat competition and extravagant methods which in earlier years had resulted in losses of millions of dollars in money and more than millions of tons of coal.

Under former conditions in the anthracite-regions, when it was not considered necessary to give thought to the morrow, and indeed up to the time when the Anthracite Coal Waste Commission made its report in 1887, it was estimated that for

every ton of coal mined and sold, 1.5 tons were lost. The greater part of this loss was in the coal left in the ground as pillars to protect the workings, while millions of tons of small coal or screenings were thrown on the culm-banks which now form unsightly mountains in the coal-regions. Improved methods of mining and of preparation have of late years reduced the percentage of waste, so that at present the recovery will average about 60 per cent. and the loss about 40 per cent. By the means of washeries, usable coal is being saved from the old culm-banks, and specially-designed furnaces have made it possible to use this fuel in steam-plants. It may also be possible in the future to recover a considerable part of the coal from the pillars in the old workings where they have not been hopelessly crushed by the settling of the overlying strata; but this could be done only at enormous expense compared with the present mining-cost, and when the burning of anthracite coal shall have become a luxury and permitted only to the wealthy. Even in our day and generation it is only by the strictest economy and skillful management in the operation of the mines that the price of coal to the consumer can be preserved as at present. The average price of anthracite at the mine ranges from $2.25 to $2.35 per long ton. What are known as "prepared sizes"—lump, broken, furnace, egg, stove, and chestnut—range from $3 to $3.75, and all the profit must be made on these. Pea and smaller sizes are sold at less than the cost of production, some as low as from 40 to 50 cents a ton. A careful study of conditions in the anthracite-region will convince the most skeptical that no robbery of the public is now being carried on.

The securing of the close control or practical monopoly that exists in the anthracite-region of Pennsylvania has been made possible by the comparatively limited area of the fields. The total area is less than 500 sq. miles and conditions are ideal for a natural monopoly. It is different in the bituminous fields, which are scattered over 30 different States and Territories. These fields aggregate about 250,000 sq. miles in area, exclusive of approximately equal areas of lower-grade coals and lignites, and are for the most part easy of access, and do not require a very large amount of capital to develop a mine. A few thousand dollars is all that is needed at first, and as

there are no restrictions on the development of new properties, every one owning land underlain by workable coal seems impelled to get it out, whether there is a profitable market for it or not. The United States produced in 1907, the banner year of industrial activity in the United States, nearly 400,000,000 tons of bituminous coal, including about 8,250,000 tons of sub-bituminous coal and lignite. In 1908, in spite of the business depression, the production was 332,500,000 tons. We could produce from the mines already open 600,000,000 tons, and we would not have to operate the mines on Sundays and holidays to do it. If the railroads supply the cars and the motive-power the mines will supply the coal. During 1907 there was, until the panic started in October, a widespread demand for transportation-facilities which the railroad companies were unable to furnish. Complaints of car-shortage came from practically every important coal-producing district. The transportation interests were subjected to all sorts of condemnation for failure to serve their patrons and the public, and while I am by no means a defender of the railroad companies (particularly since the Hepburn bill went into effect), it is fair to state that the inadequacy of the car-supply was in reality beneficial. It is doubtful if the markets could have absorbed, even in the phenomenal activity of 1907, 5,000,000 more tons of bituminous coal, only 1.25 per cent. more than the production. An increase of 5 per cent. in the car-supply and in the production would unquestionably have created a surplus and resulted in a general demoralization in values. Notwithstanding these conditions, which are fairly well known among the coal-men, new mining companies are constantly being formed and new properties opened up. The railroad companies are called upon to furnish additional switches and spurs and to provide more cars or to spread out even more thinly an already insufficient supply. As common carriers these companies cannot discriminate, and when called upon must furnish the transportation without favor. Each new mine thus opened calls for miners to work it, and miners, who are as a class nomadic, are inclined to seek employment in the newer mines. This condition reduces the supply of labor and curtails the productive capacity of the older mines; and as reduction of output means increased cost in operation, it would appear that many of these

must close down as unprofitable before all of the coal that should be extracted is won.

Under our system of government and of control over mining-operations any effective way of curbing the tendency on the part of coal-land owners to develop their properties or of protecting capital already invested in the industry is not apparent. The spirit of rivalry that exists throughout the coal-producing regions, district competing against district, and State against State, makes it useless to hope that State legislatures will place any restrictions upon the industry which will discourage further development. Yet every new mine opened has its influence on the creation of a surplus, which, while it may seem desirable to those who clamor for cheaper coal, is ultimately destructive of industry, lowers wages, and makes necessary the practice of economies (?) that are prejudicial to safety to life and property in the operation of the mines.

The year 1907, if not the most prosperous year in the history of bituminous-coal mining, was one of the most prosperous years. Production reached its maximum, and prices were the highest in recent years. Yet there were very few districts in which the margin between the cost of putting the coal on the railroad cars and the price at which it was sold was as much as 10 cents a ton. In many States it was considerably less than 10 cents, and this margin must cover such losses as are due to explosions and other accidents, indemnities paid to employees or their heirs, and all extraordinary expenses. One such explosion as that at Monongah, W. Va., in December, 1907, will wipe out many years' profits. In 1908 not only was the margin of profit much reduced in all the coal-mining districts, but thousands and hundreds of thousands of tons were sold at less than the cost of production. Of course, it is poor business to continue production at a loss, but a coal-mine is not a factory nor a quarry. To close down a coal-mine costs money. The mine must be kept clear of water; if the ventilation is stopped, gas accumulates; falls of roof and coal occur; and after a period of idleness much repair-work has to be done before operations can be resumed. It is often less expensive in the long run to continue the production of coal at a loss than to close down the mine.

It is, perhaps, somewhat bold to suggest that the bituminous mines should be put under some sort of government control,

but if they are not, I am frankly of the opinion that before many decades have passed the protection of capital already invested will make it necessary to secure control by private enterprise of certainly the areas containing the higher grades of coal and to regulate the production according to market requirements. Under our system of government the Federal authorities have no jurisdiction over mines in the several States unless the power given them under the Constitution to regulate commerce between the States could be stretched to apply to coal because of its bearing on interstate traffic. But it does look as if a choice will have to be made from three evils. The first of these is the continuation of the conditions as they now exist—a feasting for to-day and remorse for the morrow. The second is the ultimate control by a combination of interests that will make the "hard-coal trust" appear insignificant—to look "like thirty cents," as expressed in the vernacular, and the "water-power trust" would be of still less importance. The third is governmental supervision and regulation—not ownership, however. The first will be bad, the second worse; the third is problematical. Under such government control bituminous-coal mining could be regulated through a system of license; and in order that restriction on coal-production may be secured, no license should issue for the opening of a new mine until ample proof is shown that the necessities of the people or of trade require it.

I do not believe that present conditions should continue—they certainly must not continue if our coal-supplies are going to be considered—nor do I believe that it is the part of wisdom to permit the bituminous coal-supplies to get into the control of a comparatively few men living in New York and Chicago. It may be suggested that control by the several States is a fourth and best alternative. Under the competitive conditions to which I have referred it is not to be hoped or expected that the States will undertake to restrict developments in their respective jurisdictions any more than they will enact legislation which will restrict the miner in his personal liberty.

And speaking of the personal liberty of the miner, it is well known that not the least difficulty experienced in carrying on a coal-mining operation is the enforcement of discipline among the employees.

When humanity is shocked by the occurrence of some great disaster in a coal-mine, sympathy is poured out to the miners and invectives hurled against the mine-owners. He is without a soul who would withhold sympathy at such a time, but scarcely less brutal is he who holds up to the condemnation of the world the ones in authority who have by all human endeavor striven to prevent the catastrophe. It is unfortunately true that the death-record in the coal-mines of the United States shows unfavorable comparison with other countries, but it cannot be truly said that the blame should attach to the operators alone. In the great majority of cases they who suffer death or injury in the coal-mines are victims of their own carelessness, or that of their fellow-employees. The year 1907, the one of greatest production in our history, was the darkest in regard to casualties, the death-list exceeding 3,000. At one time an epidemic of explosions seemed to exist, and scarcely had the echoes of one died away before another occurred. The victims from this cause—that is, from explosions alone—numbered nearly 1,000, or approximately one-third of the total number of men killed. The statistics show, however, that more than that number were killed by falls of roof, most of which are preventable if proper precautions are taken by the men, or if, in fact, they obey the rules of the companies. In ordinary years the majority of accidents are due to roof-falls or to other preventable causes, but these occur singly and are not chronicled in the news dispatches. Even in the case of explosions, the cause may usually be traced, if any witnesses are alive to testify, to an act of carelessness or disobedience.

A prolific cause of mine-explosions is what is known as a "windy shot," due to an improperly-prepared blast, or to the failure on the part of the miner to undercut his coal, depending, as he frequently does, on the powder to do his work for him. Here it is that the strength of the mine-workers' union might be exercised for good, but unfortunately, instead of helping to secure legislation which will hold miners criminally responsible for acts of carelessness or insubordination that may result in loss of life or damage to property, the miners' influence is exerted against it. Such legislation means a restriction of their liberties as American citizens. If in the effort to enforce discipline a mine-employee is discharged for infraction

of rules, the result is, in the majority of cases, the precipitation of a strike, and the mine is laid idle for several days at least. Coal-mining is at best a hazardous occupation, and there is no line of industry in which military discipline is so essential, except perhaps in the passenger service of railroads and steamships. In European countries, where fewer accidents occur, the operations are under strict police surveillance. Both miners and operators are made to obey the laws. When this is done in the United States accidents will decrease, but the expense of mining will be increased and the price of coal will advance. On behalf of the mine-owners, it should be admitted that self-interest, if nothing else, compels the exercise of precautions against accidents. If they have no interest in securing the safety of their employees, they have at least a desire to protect their own properties. There are instances, it is true, where operators, like the men, take chances, where false economies are practiced, and where even ordinary precautions are not observed, but these, I honestly believe, are rare, very rare, exceptions.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Preparing and Recording Samples for Use in Technical Assay-Laboratories.

BY LOUIS D. HUNTOON,* NEW HAVEN, CONN.

(Spokane Meeting, September, 1909.)

AFTER the completion, in 1905, of the Hammond Mining and Metallurgical Laboratory of the Sheffield Scientific School, Yale University, it became necessary to secure and assay a large assortment of ore-samples, and to arrange the results in such a manner as to admit of use with the least amount of work on the part of the instructor.

At first, pulp-samples, together with their respective assays, were secured from the mines, smelters, and public assayers, but this practice did not prove satisfactory. It was not only difficult to secure a sufficiently large supply of material, but it was also impossible to secure the proper assortment required for instructing students. It involved considerable work for the donors to select the samples and furnish a record of the assays, and also for the instructors to examine each pulp and determine its character. Some of the samples did not contain a sufficient quantity of pulp for the student to check his work in case his first assay did not correspond to the record.

In order to increase the number of samples and to vary the character of the material, pulp-samples were later mixed with barren rock and with sulphides, but this method also was unsatisfactory and has been discontinued.

At present, small lots of ore are secured by gift or purchase from engineers, public assayers, and the managers of mines, mills, and smelters. These lots weigh from 10 to 100 lb. each, and are rarely finer than 0.25 in. in size. The larger samples permit of a careful study by the instructor in order to determine the best method of assaying to be followed.

In universities in which assaying is taught the record of the samples should fulfill the following requirements:

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1. Ease of record, avoiding all unnecessary duplication of work, and, so far as practicable, the possibility of error from duplicating numbers or in recording the assay.

2. Filing of assay-records properly, especially if the card containing the record has been taken from the file for comparison.

3. Facility in securing the character of sample desired.

4. No duplicating of numbers for the same sample.

5. The number of the last sample recorded should be easy of reference.

6. The number on the sample-sack should not in any way indicate to the student the character of the pulp contained.

7. Finding promptly the assay-record of a sample by having only the number on the sack.

8. The total number of different samples of any character should always be known, so as to replenish the stock when necessary.

The Dewey decimal system of record was adopted so as to indicate: 1, whether the sample is an ore or a furnace-product; 2, the character of the gangue; 3, for what metals the sample is to be assayed; and 4, the method of assaying to be used by the student. This system has been in use at the Hammond Laboratory for four years, and with a few slight changes has proved entirely satisfactory.

The records of the assays are entered on cards, 5 by 8 in. in size, indexed as shown in Fig. 1. Numbers in the left-hand column on the key are placed to the left of the decimal point on the assay-cards and indicate whether the sample is an ore or a furnace-product, and also for what metals the sample is to be assayed. Thus, 23 indicates gold and silver; 13, lead and silver; 83, a scale-ore of silver, such as ores from Cobalt; 4123, a lead-bullion containing gold and silver; and 913, a special furnace-product containing lead and silver.

The second set of numbers are placed to the right of the decimal point and indicate the character of the gangue and the method to be employed in assaying; thus, .13 indicates the gangue to be a basic oxide and may be either limestone or an oxide of one of the base metals; .14, the ore has a siliceous gangue which requires a basic charge with the addition of reducing flux; .5, the ore is a sulphide, 1 g. of which will reduce

more than 2.5 g. of lead, and should be run by the crucible method with the addition of nails; .6, the ore is a sulphide, 1 g. of which will reduce between 1.3 and 2.5 g. of lead, and can be run by the crucible method with the addition of niter; .7, the scorification-assay will give higher results, otherwise it is an impure ore giving a matte or speiss by the crucible method. The decimals .57 and .67 indicate that the gold is

KEY.

ORES.		GANGUE.		
1	Lead.	.1	Oxide.	
2	Gold.	.2	Sulphide.	
3	Silver.	.3	Basic.	
PRODUCTS.		.4	Acid.	
4	Bullion.	METHOD.		
5	Speiss.	.5	Nail.	
6	Matte.	.6	Niter.	
7	Cu bar.	.7	Scorification.	
SPECIAL.		.8		
8	Scale ore.	.57	Nail for gold; scorification for silver.	
9	Special.	.67	Nitre for gold; scorification for silver.	
10				
11	To be assayed.			

FIG. 1.—KEY TO CLASSIFICATION OF ASSAYS.

to be determined by a crucible charge, using the nail or niter method respectively, and the silver by scorification.

The index- and assay-cards are contained in ordinary filing-cases. One file contains the records of the common ores and another the records of the furnace-products, and ores and products requiring special treatment. The first index-card to the left, containing only the number to the left of the decimal, is of one color, and the remainder of the index-cards, indicating both the content and the character of the sample, are of another color. Cards showing the same character of sample and method of

assaying are placed in the same relative position for each set of ores, an arrangement which greatly assists in the finding of a desired sample.

The assay-card, shown in Fig. 2, contains a complete record of the sample, entered from the instructor's assay-certificate, shown in Fig. 3. The series number in the upper left-hand corner of

SERIES 23.67	Au. 2.31	Pb. 5 % ±	FeS_2 25 % ±		Samples Nos.
	Au. 36.20	Cu. 2 % ±			1492 to 1507

CHARACTER *Screen Test—10–20—Calumet.* Assayer *Jones & Brown.*

1492	1493	1494	~~1495~~	1496	1497	1498	1499
Checked by							
Jones.			*Smith.*	*Asher.*		*White.*	
$\frac{2.30}{36.27}$			$\frac{2.24}{35.80}$	$\frac{2.28}{36.10}$		$\frac{2.36}{35.70}$	
√			√	√		√	

~~1500~~	1501	~~1502~~	1503	1504	1505	1506	~~1507~~
Checked by							
Brown.		*Jackson.*	*Moore.*				*Macy.*
$\frac{2.32}{36.13}$		$\frac{1.97}{30.10}$	$\frac{2.35}{35.80}$				$\frac{2.26}{34.90}$
		√	√				√

6

FIG. 2.—RECORD OF ASSAY.

the assay-card, 23.67, indicates that the ore contains gold and silver, the gold to be determined by the crucible-method with the addition of niter, and the silver by the scorification-method. The number in the lower right-hand corner indicates that the card is the sixth in position under the index-number, 23.67. At the top of the card are entered also the results of the fire-assay, the estimated percentage of base metals determined by vanning, any note of special interest under "character," and

LABORATORY ASSAY.

Character of Gangue.	.13—Acid .14—Basic.... ....Fe........, Ca........, .2—Sulphide Fe 25%.... Pb 5%...... Cu 2% ±... As Sb........,
Series 23.67.	PRELIMINARY ASSAY. 1 Gm. Ore { Oxidizes... { Reduces.........2.................................Gms. Pb.
Assay Charge.	Ore.........½.......................½...................... .2...........................AT PbO........20......................60................ ....Gms Pb...........2 *nails*..60.................... " Niter.................................4.. " SiO_2...*B & Gl* 2.......... " Cover......*B & Gl*................*B & Gl*...................................... "

Time in furnace...50............60..........50

Slag: Color...*Black*........*Green*......... Character...*Basic*........*Glassy*.........

Button: weight...18.........20.........22 Character...*soft*...........................

....

ASSAY AND METHOD.

Litharge .1		Nails .25			Niter .26		Scorification .27	
2. Au.	3. Ag.	1. Pb.	2. Au.	3. Ag.	2. Au.	3. Ag.	2. Au.	3. Ag.
			1.12	16.30	1.14	17.50	.42	7.20
			1.10	16.50	1.16	17.20	.44	7.30
			2.22	32.80	2.30	34.70	.41	7.24
							1.27	21.74
							5	5
23.67							3) 6.35	108.70
							2.12	36.27

No. 1492......Date...3/21/07..........................Assayer......*Jones*.........................

FIG. 3.—INSTRUCTOR'S ASSAY-CERTIFICATE.

the names of the assayers. The cards are ruled to contain 8 samples, but in case there are 16 duplicate samples these are usually recorded on the same card. The first samples recorded on each line are assayed by different instructors. The names of the students are entered under the number of the sample

assigned to them, and when the samples are returned the series-number and students' assays are entered on the pulp-sack and later checked on the assay-card as having been returned. If the sack returned contains too little pulp to be of value, it is destroyed and the number on the assay-card crossed out. If the returned sack contains sufficient pulp to be of value the number is checked and the sample filed. When all of the samples recorded on one card have been given out and are also returned, the pulps are re-mixed and divided into new samples, and new numbers are given to each one, the record being entered on a new card. One sample of each lot of mixed pulps is assayed by an instructor so as to avoid the possibility of error arising from a student placing the pulp in a wrong sack.

Duplicate samples are given out under different numbers, and rarely does any one class of students receive more than two samples from the same card. The samples, as mixed, are numbered consecutively, regardless of the contents. Card 6, series 23.67 (Fig. 2), records pulps 1492 to 1507; the following card, 7, of the same series, may record pulps 6001 to 6016. Pulp 1507 is in series 23.67 and the following number, 1508, may be in series 1.5. By this method the number on the sack does not indicate to the student the character of the sample or the method to be employed in assaying.

In case it is desirable to find the assay of any special number, the series of which is not recorded on the sack, reference is made to the location-card, Fig. 4, which contains the numbers of the samples and the series to which they belong.

For instruction-work it is desirable to have as large an assortment of samples as possible with different assays. At present there are recorded and ready for use in the Hammond laboratory about 7,000 samples. The annual consumption amounts to about 2,000 samples, and the new ones added to the list each year amount to about 3,000, which gives a gain of about 1,000 samples per year. It is hoped in the near future to have on record 25,000 samples, which should afford a sufficiently large assortment of assays requiring different methods of treatment. In order to obtain this variety, products are used from the testing of ores, from screen-tests of 25-lb. lots, from mixing 5-lb. lots of comparatively coarse gold-ore with different quantities of pyrite, and silver-ore with different amounts of galena. The samples thus obtained are pulverized in jar-

mills and passed through an 80-mesh screen. For 16 gold-pulp samples, weighing about 125 g. or 4 A. T. each, 2,000 g. of pulp is taken, thoroughly mixed on a rubber sheet, and riffled into two lots, marked A and B. These lots are separately mixed and riffled into eight samples each, mixing thoroughly between each riffling, and placed in numbered pulp-sample bags. The

7233
7528

7233		~~7285~~	3.5	7303	2.6	7363		7427	
to	2.14	~~7286~~	3.5	7311		7364	123.57	to	2.14
7248		7287	32.7			7365		7443	
7249		7288		~~7312~~		7366	23.5	7444	
to	2.5	7289	2.14	to	3.7			to	2.14
7264		7290		~~7328~~				7460	
7265	1.5	7291	2.5	7329		7367		7461	
7266	1.5	7292		to	623.	to	13.7	to	3.7
7267	2.7	7293		7336		7375		7469	
7268	23.14	7294	23.5	7337		7376		~~7470~~	
		7295			23.67	to	3.5	to	2.7
		7296	13.5	7353		7392		~~7478~~	
7269		7297		7354		~~7393~~		7479	
to	123.14	7298	62		123.13	to	2.6	to	13.7
7275		7299	62	7362		~~7417~~		7495	
~~7276~~		7300	83			7418		7496	
	23.57	7301	83			to	1.5	to	2.14
~~7284~~		7302				7426		7528	

FIG. 4.—INDEX-CARD FOR LOCATING SERIES-NUMBER OF SAMPLES.

numbers of lot A are entered on the top line and the numbers of lot B on the lower line of the assay-record cards. The cards are then filed under No. 11, to be assayed, and the samples placed in stock. The first recorded samples of each lot assayed by different instructors must check within a commercial limit before the record is accepted. In case the assays do not

check, which is rarely the case, the samples are re-mixed and re-assayed.

The weight of the pulp in the sacks varies slightly, depending on the character of ore. Ores requiring charges of 0.5 A. T. weigh about 120 g.; low-grade ores requiring charges of 1 A. T. about 220 g.; and scorification-ores about 100 g. Each sack contains sufficient ore for preliminary vanning and testing, and for two sets of assays if necessary. The student reports his results, and in case his results do not check, he does not consume unnecessary time in re-assaying without personal instruction.

On the assay-certificate, Fig. 3, used by the instructor in this work, the number of the sample is entered on the lower left-hand corner, followed by the date of the assay and the name of the assayer. The "character of the gangue" is either checked before pulverizing or determined by vanning-tests, preferably the latter, as the student determines the character of the sample by this method. The preliminary assay is made, if necessary, and the oxidizing- or reducing-power entered on the certificate. The weight of the charge used and the notes taken on the furnace-work are next entered under the heading "Assay and Method."

The above information gives the series number, which is next entered, thus completing the certificate for file and entry on the assay-card. These certificates are kept on file for future reference in case an assay is disputed.

In laying out a term's work for a class it is necessary to select a large number of different samples having the same general character and method of assaying. The entire assortment of samples of gold-ore requiring a basic charge will be found under 2.14, 23.14, and 123.14, the number .14 remaining in the same relative position in the file. In like manner gold-ores to be run by the niter method will be found under 2.6, 23.6, 23.67, 123.6, and 123.67.

Although at first sight the above system may appear to be complicated and to call for a large amount of work, in practice it has worked out very satisfactorily. The system has been evolved during the past four years, until at present there is but one entry from the assay-certificate to be made on the record-card. Two duplicate assays, as a rule, suffice for recording the assays of 16 samples.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Barometric and Temperature Conditions at the Time of Dust-Explosions in the Appalachian Coal-Mines.

BY N. H. MANNAKEE, WILLIAMSON, W. VA.

(Spokane Meeting, September, 1909.)

SINCE the publication of the paper of Mr. Scholz, The Effect of Humidity on Mine-Explosions,[1] I have undertaken a study of the meager available data of barometric and temperature conditions at times of mine-explosions, and have arranged these data in a manner which may be of interest to other students of mine-explosions.

This study covers the period from 1898 to 1909, during which an ever-increasing percentage of collieries has been ventilated by fans giving air-currents of high efficiency. Previous to 1898 this percentage was smaller than at any time since. The various States have become, from time to time, more exacting in their demands upon poorly-ventilated mines. This general improvement in ventilation throughout the Appalachian field has undoubtedly brought about a condition which is annually the cause, or a very important attendant, of the high percentage of fatalities in mine-explosions, through the daily passage of from hundreds to thousands of gallons of water, invisibly suspended in the air, carried out of the mines by the ventilating-currents during the existence of certain temperature- and humidity-conditions.

There is a wide-spread opinion that these conditions exist from November 1 to March 31; but it is more likely that they have a much wider range, which varies from year to year. Certain years show an increase or decrease of rain-fall from the mean; the daily temperature may fall far below or rise far above the mean established by a record of a period of years; and the daily humidity varies likewise. If mine-explosions are influenced by a lack of moisture in the mines, certainly it may be expected that the influence will be felt at times when the

[1] *Trans.*, xxxix., 328 to 336 (1909).

moisture falls below the safety-point, and this relative condition may appear from year to year in the months of September and October, and April and May. Since the carrying-power of air for water-vapor diminishes as the temperature falls, there will be experienced in general a reduction of moisture in the mines the moment the intake temperature falls below that of the return-air. In proportion to the extent and continuance of this difference will the mine become drier, unless some artificial means is used to supply moisture up to the saturation-point of the upcast current.

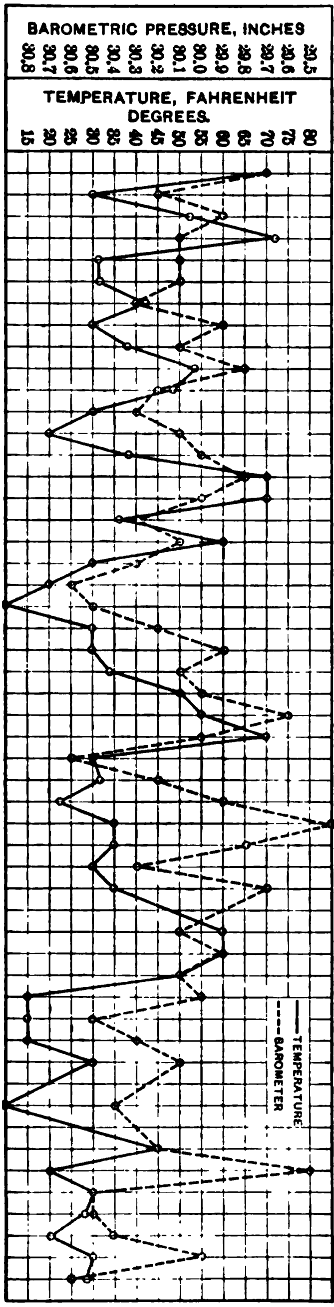
This robbing of a mine of moisture is dangerous if carried to a certain point, and this point varies under different conditions in the same locality; but the exact point at which it should be placed remains to be determined. The presence of dust in a mine, even in one portion only, has repeatedly been the cause of an explosion; and the violence of the explosion is determined by the extent of the dusty territory. The dust-explosion requires certain attendant circumstances other than the mere presence of dust, such as the initial temperature produced by a blown-out shot, the explosion of powder, etc. If these conditions could be eliminated, there would be no dust-explosions (except through the bare possibility of ignition by spontaneous combustion). They are mostly caused by ignorant, careless, or untrained miners, and we shall probably always have a certain percentage of this class of labor around the mines. So long as that is the case, the occasional direful consequences will be reaped.

I believe the real solution of the problem is, so to control the temperature and humidity of the mine-air that a dust-explosion will be impossible. I regard such control as practicable.

A careful study of the conditions can only be made after the systematic collection of data. The data herewith presented are not exact; but, so far as I can ascertain, they are sufficiently close to the truth to justify the arguments drawn therefrom. Some of them are not new, but are presented in order to make comparisons which I have not seen elsewhere.

Table I. is a complete list of mine-explosions occurring in the Appalachian coal-field from 1898 to 1909 in which five or more fatalities occurred, and which were caused by dust or gas, or both. This list is made up from the one published by

TABLE I.—*Mine-Explosions Supposed to be Caused by Gas, Dust, and Dust and Gas, Causing 5 or More Fatalities, Officially Reported in the Appalachian Coal-Field, North America,* 1898–1909 *Inclusive.*



Location.	Date.	Number of Fatalities.
Brownsville, Pa.	Sept. 23, '98	8
Mahanoy City, Pa.	Mar. 9, '99	8
Dominion Colliery No. 4, Nova Scotia.	June 16, '99	11
Grindstone, Pa.	July 24, '99	5
Sumner, Pa.	Dec. 23, '99	20
Red Ash, W. Va.	Mar. 6, '00	46
Berryburg, W. Va	Nov. 2, '00	15
Buck Mt. Colliery, Pa.	Nov. 9, '00	7
Chatham, W. Va.	May 15, '01	10
Richland, Tenn.	May 20, '01	20
Port Royal, Pa.	June 10, '01	19
Buttonwood, Pa.	Oct. 25, '01	6
Catsburg, Pa.	Mar. 6, '02	5
Dayton, Tenn.	Mar. 31, '02	16
Coal Creek (Fraterville), Tenn.	May 19, '02	184
Rolling Mill Mine, Johnstown, Pa	July 10, '02	112
Algoma, W. Va.	Sept. 16, '02	17
Stafford, W. Va.	Sept. 22, '02	6
Luke Fidler, Pa.	Nov. 29, '02	7
Ferguson, Pa.	Nov. 21, '03	17
Cheswick (Harwick Mine), Pa.	Jan. 25, '04	176
Virginia City, Ala.	Feb. 20, '05	111
Wilcox, W. Va.	Feb. 26, '05	7
Red Ash, W. Va.	Mar. 18, '05	24
Cabin Creek, W. Va.	Apr. 20, '05	6
Dubois, Pa.	Apr. 27, '05	13
Vivian, W. Va.	July 5, '05	5
Hazel Kirk, Pa.	Oct. 29, '05	5
Vivian, W. Va.	Nov. 4, '05	7
Bentleyville, Pa.	Nov. 15, '05	6
Coaldale, W. Va.	Jan. 4, '06	22
Detroit, W. Va.	Jan. 18, '06	18
Parrall, W. Va.	Feb. 8, '06	27
Piper, Ala.	Feb. 27, '06	12
Century, W. Va.[a]	Mar. 22, '06	23
Huger, W. Va.	July 19, '06	5
Pocahontas, Va.	Oct. 3, '06	36
Johnstown, Pa.	Oct. 24, '06	7
Penco, W. Va.	Jan. 26, '07	12
Fayetteville, W. Va.	Jan. 29, '07	85
Elkins, W. Va.	Feb. 4, '07	25
Whipple, W. Va.	May 1, '07	16
Naomi, Pa.[a]	Dec. '07	35
Monongah, W. Va.	Dec. 6, '07	350
Darr, Pa.[a]	Dec. '07	239
Yolande, Ala.	Dec. 16, '07	16
Port Hood, Nova Scotia	Feb. 10, '08	10
South Carrollton, Ky.	Feb. 10, '08	9
Marianna, Pa.	Nov. 28, '08	154
Lick Branch, W. Va.	Dec. 28, '08	50
Lick Branch, W. Va.	Jan. 12, '09	48
Birmingham (No. 2 Short Ck.), Ala.	Feb. 2, '09	18

Total fatalities, 2,126

[a] No data. Temperature assumed at 32°.

H. N. Eavenson,[2] and from certain other data collected personally to bring it to date. Opposite this chronologically-arranged list are recorded in curves the approximate temperature and pressure which obtained at the time of each explosion. These data were collected at the Weather Bureau at Washington, D. C., and while in most cases not absolutely accurate, may be taken as approximately so. The curve shows at a glance the range of explosion-temperatures, the highest recorded being 70° and the lowest 10° F. The barometric curve shows the range of pressure.

There are, no doubt, a number of the explosions listed in Table I. in which both gas and dust figure as causes. The questions concerning gas and dust combined are undergoing experimental investigation at the U. S. Geological Survey Technologic Bureau at Pittsburg, Pa., and much interesting information is being secured. But the question, how to control temperature and humidity of intake mine-air, is receiving minor consideration.

TABLE II.—*Barometric Condition at Times of Explosion.*

Explosions.	Number.	Per Cent.
In high-pressure areas,	23	46.94
In mean-pressure areas,	18	36.73
In low-pressure areas,	8	16.33
Total,	49	100.00

Table II. shows the barometric condition at the time of explosions. The daily weather-maps, issued by the United States Weather Bureau, show lines of equal barometric pressure; and certain points where explosions occurred were located in high-pressure or low-pressure areas, or occupied a mean between high and low pressure. Table II. shows that 9 out of 49 explosions for which data were available, 23, or 46.94 per cent., took place in high-pressure areas. The number of explosions which occurred in mean-pressure areas was 18 out of 49, or 36.73 per cent. The number of explosions occurring in low-pressure areas was 8 out of 49, or 16.33 per cent. This indicates that the pressure is generally likely to be high at the time of an explosion. Dry air is heavier than damp air, hence dry air indicates high barometer.

[2] *Bulletin No.* 30, June, 1909, pp. 567 to 578.

Table III. shows the number and percentage of explosions at certain temperatures and the number and per cent. of fatalities.

TABLE III.—*Explosions at Various Temperatures, and Number of Fatalities Reduced to Percentage.*

External Temperature. At or Below.	Number.	Per Cent.	Number of Fatalities.	Per Cent.	Number of Fatalities per Explosion.
32° F.,	29	55 77	1,539	72.39	53.07
40° F.,	36	69.23	1,658	77.98	46.06
50° F.,	41	78.85	1,721	80.95	41.9
60° F.,	47	90.39	1,812	85.23	38.6
Above 60° F.,	5	9.61	314	14.77	62.8
Total explosions considered,	52	100.00	2,126	100.00	40.9

Particular attention should be given to the explosions occurring above 60° F. Here are 5 explosions out of 52, or 9.61 per cent., and in these explosions 314 out of a total of 2,126 fatalities, or 14.77 per cent. It is my opinion that these 5 explosions, occurring above 60° F., were strictly gas-explosions. Thus far I have not had access to the printed reports of these cases, but I feel that it is not probable that dust caused them in any way. There is also a certain percentage of the explosions which occurred at 60° F. and below which are distinctly gas-explosions and in which dust plays no important part; and these should not be included in the list. There are well-defined means of regulating purely gaseous mines; and such accidents should not be included in a careful study of dust (and gas-and-dust) explosions. Table III. is, therefore, not a fair *résumé* of explosions which have in whole or in part been caused by the presence of dust. It will be possible to rearrange this table later by eliminating all explosions caused by gas alone.

Table IV., reproduced from the paper of H. N. Eavenson,[3] shows that during the winter months water is removed from the mines by the ventilating-current at the rate of from 2,777 to 16,694 gal. per 24 hr.; and that in the summer months it is deposited in the same mines at the rate of from 2,978 to 8,088 gal. per 24 hr. These figures are probably the extremes; but I think that June, July, and early August will show

[3] *Bulletin No.* 30, June, 1909, p. 568.

TABLE IV.—*Humidity-Tests at Various Mines in Southern West Virginia.*[4]

Mine No.	Date.	Outside Air.			Mine Air.			Aqueous Vapor Removed.	Quantity of Air in Circulation.	Amount of Water Removed from Mine.			
		Tempera-ture.	Saturation.	Aqueous Vapor.	Tempera-ture.	Saturation.	Aqueous Vapor.			Per 24 Hr.		Per Minute.	
		F.°	Per Cent.	Gr. Cu.Ft	F.°	Per Cent.	Gr. per Cu. Ft.	Gr. Cu. Ft. Air.	Cu. Ft. per Min	Gal.	Short Tons.	Gal.	Lb.
1....	1/24-08	21.8	44	0.591	53.5	97	4.467	3.876	130,000	12,428	51.8	8.6	72.0
1....	2/17-08	29.5	52	0.985	53.8	85	3.955	2.970	128,000	9,377	39.1	6.5	54.3
1....	2/18-08	40.5	44	1.277	53.5	84	3.868	2.591	128,000	8,131	34.1	5.7	47.4
1....	2/19-08	41.0	78	2.305	53.5	85	3.914	1.609	128,000	5,030	21.2	3.5	29.4
1....	2/20-08	28.3	66	1.186	53.8	82	3.751	2.565	128,000	8,098	33.8	5.6	46.9
2....	2/18-08	22.0	79	1.070	53.0	97	4.390	3.320	144,000	11,792	49.2	8.2	68.3
2....	2/19-08	41.8	70	2.130	52.8	98	4.405	2.275	144,000	8,031	33.7	5.6	46.8
2....	2/20-08	25.7	75	1.201	53.0	97	4.390	3.189	144,000	11,326	47.2	7.9	65.6
2....	2/21-08	27.5	76	1.319	52.8	97	4.286	2.967	144,000	10,558	43.9	7.3	61.0
2....	2/22-08	30.0	47	0.910	52.5	97	4.316	3.406	144,000	12,097	50.2	8.4	70.0
2....	2/24-08	32.8	56	1.215	52.7	98	4.390	3.175	144,000	11,277	47.0	7.8	65.3
2....	2/25-08	37.6	72	1.877	52.5	98	4.360	2.483	144,000	8,820	36.8	6.1	51.1
2....	2/26-08	35.5	83	2.156	52.6	99	4.420	2.264	144,000	8,041	33.5	5.6	46.6
6....	2/21-08	33.0	43	1.187	52.8	94	4.225	3.088	200,000	15,234	63.5	10.6	88.2
6....	2/22-08	24.5	68	1.031	50.5	97	4.024	2.993	200,000	14,765	61.6	10.2	85.5
6....	2/24-08	39.0	35	0.961	52.7	97	4.345	3.384	200,000	16,694	69.6	11.6	96.7
6....	2/25-08	29.3	56	1.051	52.4	97	4.301	3.250	200,000	16,033	66.9	9.1	92.9
2A.	Jan., 08	41.0	92	2.719	59.0	94	5.222	2.503	105,000	6,483	27.0	4.5	37.5
2A.	Feb., 08	37.0	91	2.321	58.0	94	5.048	2.727	106,000	7,130	29.7	4.9	41.3
1A.	Mar., 08	54.0	76	3.561	56.0	98	4.665	1.104	102,000	2,777	11.6	1.9	16.1
								Aqueous Vapor Deposited.		Amount of Water Deposited in Mine.			
								Gr.Cu. Ft.Air.					
6....	8/19-08	83.5	57	6.938	60.0	99	5.687	1.251	150,000	4,629	19.3	3.2	26.8
6....	8/20-08	71.0	80	6.592	60.0	99	5.687	0.905	150,000	3,348	14.0	2.3	19.4
6....	8/22-08	72.0	93	7.912	60.2	99	5.726	2.186	150,000	8,088	33.7	5.6	46.8
4....	8/24-08	80.0	58	6.842	59.2	99	5.437	0.806	150,000	2,978	12.4	2.1	17.2

larger quantities of water deposited. There is a gradual passage, step by step, from the condition of robbing the mines of moisture in winter to the deposition of moisture in summer; and there is a time at which the ventilating-current neither gives nor takes moisture in traveling through the mine. I believe that this balanced condition occurs in southern West Virginia from time to time, between the last of March and the middle of May, alternating between short periods when the mines are robbed of moisture and periods when they receive moisture from the ventilating-current. These fluctuations occur until about the middle of May, after which the summer condition very likely prevails until the middle of September, when we again experience the variation between excess and scarcity

[4] *Bulletin No.* 30, June, 1909, p. 568.

of moisture. Then, in the last part of October or the early part of November, the winter condition is installed again, and the mines give up their daily quota of water to the ventilating-current. This opinion is by no means established by complete evidence; yet a comparison of the daily temperature-curves at a particular place tends to confirm it. Since the temperature of the intake air indicates to a large extent the quantity of moisture which it carries, we can assume for practical purposes that when the outside temperature is below the mine-temperature the mine will give up moisture to the ventilating-current.

Table IV. shows that the degree of saturation of mine-air varies from 82 to 99, while that of the outside air varies from 35 to 93 per cent. The percentage of saturation of the outside air in winter is evidently less than in summer, and the same condition exists in regard to the mine-air, but is not so pronounced. Considering the saturation of the outside air at different temperatures, Table IV. indicates that the lower the temperature the smaller the percentage of saturation. And since the carrying-power of the air diminishes as the temperature falls, it would be expected that the quantity of moisture carried out of a mine in 24 hr. will increase in greater ratio than the difference in temperature between the intake and the effluent air. In other words, there are two controlling conditions:

1. Lower temperature usually means lower percentage of saturation; and

2. Lower temperature indicates lower carrying-power for moisture.

On the other hand, increase of temperature indicates a higher percentage of saturation, and higher temperature indicates higher carrying-power for moisture. Thus it would seem that when the temperature of the intake air is 5° higher than the temperature of the effluent air, a certain quantity of water would be deposited in the mine, and when the temperature of the intake air is 5° lower than that of the effluent air a much larger quantity of water would be carried out of the mine. It naturally follows that at those times of the year when the outside temperature falls several degrees below the inside temperature the mines will become drier. The greater this difference in temperature of intake air and effluent air, and the

longer this condition exists, the more pronounced will become this dry condition. All of those mines which become dusty and which are liable to be dangerous on this account have a certain safety-point which lies somewhere between the extremes pointed out above. There are no definite records to show at what temperature certain mines begin to become dry, or when they will become wet. Yet these two questions are on the very threshold of the inquiry.

In order to facilitate the thorough investigation of this phase of mine-explosions, I recommend that the following data be recorded daily at each mine employing a certain number of men in the Appalachian coal-field:

1. Temperature of outside air (continuous record).
2. Temperature of inside air (continuous record).
3. Humidity of outside air (continuous record or several readings daily).
4. Humidity of inside air (continuous record or several readings daily).
5. Volume of air in circulation.
6. Number of hours in circulation.

These data, accumulated through a period of months, preferably years, would, I believe, shed light on this serious problem and would bring us nearer to a knowledge of the effect of humidity in mine-explosions. By this means of collecting data the fluctuation in the moisture-content of the mine-air could be ascertained, and the amount of water carried out of any particular mine or deposited therein by the ventilating-current for any period could be determined. Then, when dust-explosions occurred, reference could be made to the daily record of conditions at the mines, and from this starting-point the probable cause could be sought.

So long as it is not known at just what point of relative dryness the condition of a mine becomes dangerous, we shall continue to follow along the same old rut, or have the law-makers force on all mines rigid laws which are mere guesses at the real remedy. The first course will continue the present increasing toll in human lives, and the second is likely to do the same, besides forcing on the operators the practically useless expenditure of large amounts of money. This is economic waste, and is particularly unfair to the small operator.

The conditions in different localities vary. Legislation tends to take on a compulsory uniform form, and many mines are therefore forced to comply with regulations which are useless. The mine which is operated by shaft generally has to contend with conditions entirely different from those in the same seam where it is located above water-level and high in a mountain. One is likely to generate gas and become very dry, while the other rarely develops gas and is seldom dangerously dry. These are the two extremes; and as the seam goes under heavier and heavier cover and the area of solid coal increases, the conditions of shaft-mining will probably be more nearly approached. Certainly, when the truth is known, the operation of mines will be regulated in proportion as they are more or less dangerous. Some excellent laws already exist, but a problem is now confronted which defies solution. Possibly this very winter direful explosions may occur close together, and create such a popular storm of sympathetic but ignorant indignation that some very drastic though futile mine-laws may be enacted. To prevent such hasty action it is necessary to have recorded facts. The daily records here suggested would establish very closely for each particular locality the times of the year at which the pendulum of moisture swings to the side where the mines are robbed of moisture, and *vice versâ*. This knowledge could be applied to those mines in which it is evident that dry conditions are apt to reach the danger-point.

What temperature should be selected as perilous? In Table IV. it will be noted that the temperature of the mine-air varies from 50.5 in mid-winter to 60.2° F. in almost mid-summer. It is also to be noted that one reading of mine-air in January, 1908, showed a mine-temperature of 59° with an outside temperature of 41°. I will assume in this particular case that had the outside temperature been raised to 60° the inside temperature would have risen at least to the same point (60°). Now from Table IV. it is evident that with the same outside and inside temperatures at the mine under consideration, the ventilating-currents would rob the mine of moisture, since the per cent. of saturation of outside air is uniformly less than the per cent. of saturation of inside air. This assumption may not be justified when we have records of many readings, but the indications in its favor are strong. Furthermore, these humidity-

conditions may not be general throughout the Appalachian coal-field, but since they are the only ones for which data are available, let us assume, for the present purpose, that they represent average conditions. With this explanation, let us further assume that 60° outside temperature is that temperature which if reached will assure the robbing of the mines of moisture. This temperature will not cause a mine to become dry rapidly, but if a mine has been dry all winter and the temperature of 60° is reached in the spring, and if the temperature of any dust in a particular mine were raised to the ignition-point, an explosion would naturally result, the force or extent of which would be limited only to the dusty area in the mine; and if gas were present the violence would be increased. Had this condition of temperature prevailed in an excessively dry fall, in September or October, the same result would likely have been the case.

Assuming 60° as our critical temperature when a dry condition exists in the mines, let us consider Table III. This table shows that the total number of explosions in the last 10 years was 52, and of these 47 occurred either at 60° or below. This represents 90 per cent. of all explosions from all causes. Considering fatalities, 1,812 out of 2,126, or 85.23 per cent. of the total, were caused either at or below this temperature.

If we were able to eliminate all of those explosions which were due to gas, pure and simple, and not affected by dust, the figures would be far more conclusive. I have previously pointed out that I consider all of those explosions which occurred above the temperature of 60° F. as gas-explosions and not influenced by dust. The indication is that during the period under consideration we have not experienced a dust-explosion when the outside temperature was above 60° F. Close study of data (to be collected) may show that the temperature may fall from 5° to 10° below this point before danger of dust-explosions will be experienced.

TABLE V.—*Average Number of Fatalities for Certain Ranges in Temperature.*

Range in Temperature. Degrees Fahrenheit.	Average Number of Fatalities.
Up to 60°	38.6
Up to 50°	41.9
Up to 40°	46.06
Up to 32°	53.09

Table V. shows that as the temperature falls below 60° the average number of fatalities per explosion increases.

TABLE VI.—*Explosions at Various Ranges of Temperature.*

Range of Temperature. Degrees Fahrenheit.	Explosions. No.	Explosions. Per Cent.	Fatalities. No.	Fatalities. Per Cent.	Number Fatalities Per Explosion.
10 to 20	9	17.31	740	34.80	82.22
20 to 30	14	26.92	429	20.18	30.64
30 to 40	13	25.00	489	23.00	37.62
40 to 50	5	9.62	63	2.97	12.60
50 to 60	6	11.54	91	4.28	15.16
60 to 70	5	9.61	314	14.77	60.30
Total, 10° to 70°	52	100.00	2,126	100.00	40.90

Table VI. shows that as the temperature falls below 40° the number of fatalities increases greatly. It is my opinion that the increased number of fatalities listed between the temperatures of 50° and 60° is caused by the fact that there are included certain gas-explosions which swell the total. Even so, the data indicate that 60° of outside temperature is dangerous. The cause of this increase in number of fatalities as the temperature falls, naturally follows from the fact that mines become drier as the temperature falls, and the extent and violence of a particular explosion vary in some way directly as the dry area in the mine. The presence of gas, of course, will increase the violence. But back of all is the question of humidity, which is so closely linked with temperature in its rise and fall.

There must be a practical solution to this problem, and I believe it can be found. The method of spraying by means of a car, or by means of pipe-lines with sprays located at intervals, I regard as unsatisfactory and wasteful of water in the first case, and very expensive and cumbersome in the latter. The use of calcium chloride, $CaCl_2$, as a deliquescent is excellent for haulage-ways and partings, but it will prove very expensive if used in all working-places, in which most of the dust is generated and where the greatest danger from explosion lies. Whitewashing, and the use of slate- and shale-dust as a damper for explosive conditions, of course possess merit, but these are local preventive measures, and require too much time and supervision to be followed systematically.

I am inclined to the view that it will be necessary to install a blower-system such as is used in ventilating large office-

buildings, which heats the air to the required temperature. The air, having been heated to a temperature equal to the critical temperature, must be saturated with moisture at that point. This saturation would best be done near the intake, inside the mine. Long shallow pans might be tried, or a series of sprays arranged so that the current of air would be forced through the finely-divided water. The number of flat shallow pans or the number of sprays needed could readily be determined by experiment. This method of control would be less expensive, and more economical in the consumption of water, than the intricate watering-systems which are laid in each entry of the whole mine, and being concentrated in a small area it could be regulated automatically. It is not my intention to describe this system in detail here, but I may do so at a later time if the subject proves of sufficient interest.

The installation of such a system in dangerously dusty mines would reduce explosions from dust to a minimum. It may be found that certain portions of mines will still be persistently dusty, just as we find such spots, even in midsummer. These troublesome places can be dampened by a spraying-car or by some other local means of application. Moreover, in all cases and with whatever auxiliaries, that system of mining the coal should be practiced which will produce the minimum quantity of dust, and the method of firing and the explosives used should be determined by considerations of safety.

Conclusions.

1. There is a certain condition of the outside air, having a temperature and humidity such that when it is used in ventilating mines moisture will neither be deposited in the mines nor carried out of them. This condition I call the "critical condition." And this condition probably varies slightly according to latitude, altitude, and the seasons of the year. This range of variation will probably fall between temperatures of 50° and 60°.

2. As the temperature of the outside air falls below the critical point, mines will be robbed of moisture by the ventilating-current (and *vice versâ*).

3. While this lower outside temperature continues, the mine will become drier, and in proportion to the difference between

this lower outside temperature and the critical point, the rate of drying will be increased or diminished.

4. When the outside temperature is such as to make the ventilating-current dry the mine, this effect will be increased by increasing the volume of the current.

5. This critical condition should be determined, and a ventilating-and-heating system should be adopted which would raise the temperature of the intake air to the mine-temperature, and then saturate the air with moisture.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

A Method of Calculating Sinking-Funds, and a Table of Values for Ordinary Periods and Rates of Interest.

BY J. B. DILWORTH, PHILADELPHIA, PA.

(—— —— Meeting, February, 1910.)

In estimating the investment-value of a mining-property or plant, the value of which decreases with operation, it is often necessary to know the sum which must be set aside periodically from earnings either to renew the plant at the end of its life, or to return the capital involved when the earning-power of the investment has ceased. In such cases it is manifestly incorrect simply to divide the total amount to be retired by the number of periods (usually years) during which the investment shall be active, as the sums periodically withdrawn have a certain interest-earning power varying with the length of time they are held and with the opportunities afforded for their investment.

But though the interest may be considered to accumulate regularly year by year, yet the rules and tables for compound interest do not apply, owing to the addition of a fixed amount of new money to the total at the beginning of each period.

Briefly stated, the problem of sinking-funds is, to find the amount of money which must be periodically set aside at a certain rate of interest, regularly compounded, to yield a certain sum in a given length of time. In most practical cases the interest is compounded and the fresh amount added at the same time—usually once a year—and on this basis the problem may be solved as follows:

Let S = total amount to be retired,
Let r = interest rate: $R = 1 + r$,
Let n = life of the sinking-fund or the amortization-period,
Let x = the sum set aside at the end of each year.

Then the amount of the sinking-fund at the end of first year is x, which at the end of n years will have increased by compound interest to xR^{n-1}. Similarly, another amount, x, will be set aside at the end of second year, which will in turn increase

to xR^{n-2} at the end of the amortization-period; and so on to the last payment at end of the nth year, when the amount will be simply x. Therefore, the total amount, S, at the end of the amortization-period, will be

$$S = xR^{n-1} + xR^{n-2} + xR^{n-3} \quad . \quad . \quad . \quad . \quad + xR^3 + xR^2 + xR + x.$$

This equation is reducible to simpler form by multiplying through by R and subtracting the first equation from the result, thus:

$$\begin{array}{rl} RS = & xR^n + xR^{n-1} + xR^{n-2} \quad . \quad . \quad . \quad . \quad + xR_4 + xR^3 + xR^2 + xR \\ S = & \qquad\quad xR^{n-1} + xR^{n-2} \quad . \quad . \quad . \quad . \quad + xR^4 + xR^3 + xR^2 + xR + x \\ \hline RS - S = & xR^n \qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad -x, \text{ or} \end{array}$$

$$S\,(R-1) = x\,(R^n - 1), \text{ or } x = S\frac{R-1}{R^n-1} = S\frac{r}{(1+r)^n-1}.$$

As an illustration of the method of using this formula, let it be assumed that the amount to be retired, S, is \$70.95; the interest rate, r, 5 per cent., and the amortization-period, n, 16 years. Substituting in the formula, $x = 70.95\,\dfrac{0.05}{(1.05)^{16}-1}$ $= 3$, *i.e.*, \$3 set aside at the end of each year at 5 per cent. interest, annually compounded, would amount in 16 years to \$70.95.

Results obtained with the above formula may be checked by finding by actual multiplication and addition what \$1 set aside each year will amount to in the given time at the assumed rate of interest. Then a simple proportion will give the sum which must be annually invested at the same interest rate to furnish the required sum at the end of the amortization-period. In the foregoing example it may be found arithmetically that a \$1 a year fund, interest 5 per cent., will amount in 16 years to \$23.65. Now if \$23.65 is produced by a \$1 a year fund, \$70.95 will be produced by a \$3 fund, or, expressed in the form of a proportion, 23.65 : 1 : : 70.95 : 3.

If payments are to be made and interest compounded semi-annually then

$$\tfrac{1}{2}\,x = S\frac{\frac{1}{2}\,r}{(1+\frac{1}{2}\,r)^{2n}-1}, \text{ or } x = S\frac{r}{(1+\frac{1}{2}\,r)^{2n}-1}.$$

Table I. gives the amount which a \$1 a year fund will afford in from 1 to 50 years with interest-rates varying from 2 to 8 per cent. By the method of proportion just illustrated these

results may be applied to any case in hand. No originality is claimed in presenting this table—more extensive ones may be found in the files of savings banks and insurance companies—but it is given here in the belief that it may serve some to whom the more complete ones are not readily accessible.

TABLE I.—*Sinking-Fund Table.*

Time: At End of Year.	*Rate of Interest.* 2 Per Cent.	3 Per Cent.	4 Per Cent.	5 Per Cent.	6 Per Cent.	7 Per Cent.	8 Per Cent.
1st	1.00	1.00	1.00	1.00	1.00	1.00	1.00
2d	2.02	2.03	2.04	2.05	2.06	2.07	2.08
3d	3.06	3.09	3.12	3.15	3.18	3.21	3.25
4th	4.12	4.18	4.25	4.31	4.37	4.44	4.51
5th	5.20	5.31	5.42	5.52	5.64	5.75	5.87
6th	6.31	6.47	6.63	6.80	6.98	7.15	7.34
7th	7.43	7.66	7.90	8.14	8.39	8.65	8.92
8th	8.58	8.89	9.21	9.55	9.90	10.26	10.64
9th	9.75	10.16	10.58	11.03	11.49	11.98	12.49
10th	10.95	11.46	12.01	12.57	13.18	13.82	14.49
11th	12.17	12.81	13.49	14.21	14.97	15.78	16.65
12th	13.41	14.19	15.03	15.91	16.87	17.89	18.98
13th	14.68	15.62	16.63	17.71	18.88	20.14	21.50
14th	15.97	17.09	18.29	19.60	21.01	22.55	24.22
15th	17.29	18.60	20.02	21.58	23.27	25.13	27.15
16th	18.64	20.16	21.82	23.65	25.67	27.89	30.33
17th	20.01	21.76	23.70	25.84	28.21	30.84	33.75
18th	21.41	23.42	25.66	28.13	30.90	34.00	37.45
19th	22.84	25.12	27.68	30.54	33.76	37.38	41.45
20th	24.30	26.87	29.79	33.06	36.78	41.00	45.76
21st	25.78	28.68	31.98	35.72	39.99	44.86	50.43
22d	27.30	30.54	34.26	38.50	43.39	49.01	55.46
23d	28.84	32.46	36.63	41.43	46.99	53.44	60.90
24th	30.42	34.43	39.10	44.50	50.81	58.18	66.77
25th	32.03	36.46	41.66	47.72	54.86	63.25	73.11
26th	33.67	38.56	44.33	51.11	59.15	68.68	79.96
27th	35.34	40.71	47.10	54.66	63.70	74.48	87.35
28th	37.05	42.93	49.98	58.39	68.52	80.70	95.34
29th	38.79	45.22	52.98	62.31	73.64	87.35	103.97
30th	40.57	47.58	56.10	66.43	79.05	94.46	113.29
31st	42.38	50.01	59.34	70.75	84.80	102.07	123.35
32d	44.23	52.51	62.72	75.29	90.88	110.22	134.22
33d	46.11	55.08	66.23	80.05	97.34	118.93	145.96
34th	48.03	57.73	69.88	85.05	104.18	128.26	158.63
35th	50.00	60.46	73.67	90.31	111.43	138.24	172.32
36th	51.99	63.28	77.62	95.82	119.11	148.91	187.11
37th	54.03	66.18	81.72	101.61	127.26	160.34	203.08
38th	56.11	69.16	85.99	107.69	135.90	172.56	220.33
39th	58.24	72.24	90.43	114.08	145.05	185.64	238.95
40th	60.40	75.40	95.05	120.78	154.75	199.63	259.07
41st	62.61	78.67	99.85	127.82	165.04	214.61	280.79
42d	64.86	82.03	104.84	135.21	175.94	230.63	304.26
43d	67.16	85.49	110.04	142.97	187.50	247.78	329.60
44th	69.50	89.05	115.44	151.12	199.75	266.12	356.97
45th	71.89	92.72	121.06	159.68	212.73	285.75	386.52
46th	74.33	96.51	126.90	168.66	226.50	306.75	418.44
47th	76.82	100.40	132.98	178.10	241.09	329.22	452.92
48th	79.35	104.41	139.30	188.00	256.55	353.27	490.15
49th	81.94	108.55	145.87	198.40	272.94	379.00	530.37
50th	84.58	112.80	152.70	209.32	290.32	406.54	573.80

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Chemical Laboratories in Iron- and Steel-Works.

BY GEORGE W. MAYNARD, NEW YORK, N. Y.

(—— Meeting, February, 1910.)

In the biographical notice of Thomas F. Witherbee, published in *Bulletin No.* 32, August, 1909 (p. xxv), it is said that "he is believed to have been the first manager in America to use the chemical laboratory for the purpose of controlling the regular running of the blast-furnace."

Since the year is not given, I cannot decide as to Mr. Witherbee's priority; but the statement leads me to contribute a bit of history, showing an early departure from the "rule-of-thumb" blast-furnace work to the employment of a chemist.

In the autumn of 1868 the department of mining and metallurgy was established at the Rensselaer Polytechnic Institute. In connection with the assay-department a laboratory was equipped for making general commercial analyses.

In January, 1869, Alexander L. Holley, the manager and builder of the Bessemer plant at the Rensselaer Works of John A. Griswold & Co., the first plant erected in the United States under the Bessemer patents, began to send me samples of ore, pig-iron, and steel for analysis. Early in 1869 I was regularly retained as chemist for the Rensselaer Works; and while the laboratory was not located at the works, for all practical purposes it was the works-laboratory.

The increase in work soon made it necessary for me to secure an assistant. I applied to Prof. Charles F. Chandler, who recommended Dr. August Wendel, who had been an assistant of Rammelsberg and had lately landed in New York. Dr. Wendel remained with me until 1873, when I left Troy. My leaving made necessary the erection of a laboratory at the Rensselaer Works, which was done under my supervision, and Dr. Wendel was employed as chemist.

During the period from 1868 to 1873 my work as consulting mining engineer frequently called me to the Lake Champlain

iron-district and involved the examination and sampling of about all the developed mines and prospects, in hunting for Bessemer ores; for, astonishing as the statement may seem, in the early history of the Bessemer process in the United States pig-iron was imported from the West Cumberland district in England. From 1868 to 1872 I frequently visited Mr. Witherbee at Fletcherville, and was always greatly impressed by his energy, originality, and knowledge of the best European practice. My last meeting with him, at Durango, in Mexico, was a pathetic one, for he was then almost totally blind.

On referring to my record of many hundred analyses during those years, I find that samples of ore, pig-iron, slag, and steel came from many widely-scattered works, which would indicate that chemical laboratories were exceptional in connection with iron- and steel-works.

No. 36 DECEMBER 1909

Bulletin of the American Institute of Mining Engineers.

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EDITORIAL OFFICE AT 29 WEST 39TH STREET, NEW YORK, N. Y.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.

No. 36 DECEMBER 1909

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SECTION I.—INSTITUTE ANNOUNCEMENTS.

This section contains announcements of general interest to the members of the Institute, but not always of sufficient permanent value to warrant republication in the volumes of the *Transactions*.

SECTION II.—TECHNICAL PAPERS AND DISCUSSIONS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

A detailed list of the papers contained in this section is given in the Table of Contents. They have been so printed and arranged (blank pages being left when necessary) that they can be separately removed for classified filing, or other independent use.

A small stock of separate pamphlets, duplicating the technical papers given in Section II. of this Bulletin, is reserved for those who desire extra copies of any single paper.

Comments or criticisms upon all papers given in this section, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

All communications concerning the contents of this Bulletin should be addressed to Dr. Joseph Struthers, Assistant Secretary and Editor, 29 W. 39th St., New York, N. Y. (Telephone number 4600 Bryant).

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For the year ending February, 1910.

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

INSTITUTE ANNOUNCEMENTS.

Honorary Membership.

On Nov. 11, 1909, M. Alexandre Pourcel, of Paris, France, upon the proposal of a large number of distinguished members, and the unanimous recommendation of the Council, was elected by the Board of Directors an Honorary Member of the Institute.

M. Pourcel has devoted forty-three years to the metallurgy of iron and steel; and the work which he did between thirty and forty years ago constitutes an important part of the foundation of the practice of to-day. Among his striking contributions are: (1) his perfection, in 1869, of the manufacture of high-grade ferro-manganese in a carbon-lined open-hearth furnace, instead of the costly crucibles previously used; (2) his discovery, in 1874, of the method of making solid steel castings; (3) his manufacture, in 1875, of ferro-manganese and silico-spiegel in the blast-furnace; and (4) his anticipation, in 1875, of modern American blast-furnace practice, by the production, in a very small furnace, of Bessemer pig containing 2.5 per cent. of silicon, from a burden yielding only 46 per cent. of iron, and with a consumption, per ton of pig, of only 1,840 lb. of coke, containing 15 per cent. of ash. These brilliant pioneer achievements, together with his contributions to current technical literature, won for him the Bessemer gold medal of the Iron and Steel Institute; the great "medal of honor," which is the highest award of the French Society of Mineral Industry, and other gold medals, as well as the distinction of honorary membership of the Comité des Forges de France, which was conferred upon him in 1891.

In recent years, M. Pourcel has been instrumental, as consulting engineer of the Société de Commentry-Fourchambault, at Decazeville, France, in an extraordinary development of the manufacture of special steels. The Invar plant, of this company, is said to make a greater variety of such steels (includ-

ing "platinite," "invar," and other brands) than any other in the world.

M. Pourcel has been a member of the American Institute of Mining Engineers for more than thirty years, and has made to its *Transactions* many contributions of timely significance and permanent value. Not only his valuable researches and practical achievements, but also the generosity with which he has communicated to his professional colleagues the results of his study and experience, entitle him beyond question to the recognition now given to him by his election as Honorary Member of this Institute.

Meetings of Other Societies.

American Society of Mechanical Engineers.—The thirtieth annual meeting of the American Society of Mechanical Engineers was held in the Engineering Societies Building, December 7 to 10. The Meetings Committee, Willis E. Hall, Chairman, had entire charge of the four professional sessions, which included a symposium on the Measurement of the Flow of Fluids, and sessions devoted to Steam Engineering subjects and to the Gas Power Section. Important reports of the Gas Power Section were presented for discussion.

The following papers were discussed:

Tests on a Venturi Meter for Boiler Feed, by Chas. M. Allen.
The Pitot Tube as a Steam Meter, Geo. F. Gebhardt.
Efficiency Tests of Steam Nozzles, F. H. Sibley and T. S. Kemble.
An Electric Gas Meter, C. C. Thomas.
Tan Bark as a Boiler Fuel, David M. Myers.
Cooling Towers for Steam and Gas Power Plants, J. R. Bibbins.
Governing Rolling Mill Engines, W. P. Caine.
An Experience with Leaky Vertical Fire-Tube Boilers, F. W. Dean.
The Best Form of Longitudinal Joint for Boilers, F. W. Dean.
Testing Suction Gas Producers with Koerting Ejector, C. M. Garland, A. P. Kratz.
Bituminous Gas Producers, J. R. Bibbins.
The Bucyrus Locomotive Pile Driver, Walter Ferris.
Line Shaft Efficiency, Mechanical and Economic, Henry Hess.
Pump Valves and Valve Areas, A. F. Nagle.
A Report on Cast-Iron Test Bars, A. F. Nagle.

On Wednesday afternoon there was an excursion for the in-

spection of the new Pennsylvania Railroad Station, in New York City, attended by the members and guests in a body. In addition to this general excursion, there were opportunities for smaller parties to visit other places of interest.

On Tuesday evening, President Jesse M. Smith presented his address on The Profession of Engineering, which was followed by the report of the tellers of election of officers; the introduction of the new President, George Westinghouse, of Pittsburg, Pa., and a reception by the President and President-elect in the rooms of the Society.

On Wednesday evening there was an illustrated lecture, The Development of Agricultural Machinery, by L. W. Ellis, of the Bureau of Plant Industry of the United States Agricultural Department.

On Thursday evening a reception was held at the Hotel Astor.

The following officers were elected:

President, George Westinghouse.

Vice-Presidents, Charles Whiting Baker, W. F. M. Goss, E. D. Meier.

Managers, J. Sellers Bancroft, James Hartness, H. G. Reist.

Treasurer, William H. Wiley.

American Institute of Electrical Engineers.—The two hundred and forty-first meeting of the American Institute of Electrical Engineers will be held in the auditorium of the Engineers' Building, 33 West Thirty-ninth Street, New York, on Thursday evening, Dec. 16, 1909, at 8 o'clock. This meeting will be held under the auspices of the High Tension Transmission Committee. Mr. Henry L. Doherty will present a paper, entitled, Comments on Development and Operation of Hydro-Electric Plants. In addition to the engineering features, this paper closely relates to the conservation of our natural resources, the financing and sale of securities of power projects of all kinds, and the matters of rates, penalties, and insurance of service. It should therefore appeal to those especially interested in these matters, as well as to those interested in the purely engineering aspects of hydro-electric enterprises.

Members of the American Institute of Mining Engineers are cordially invited to attend these meetings.

International Congress for Mining, Metallurgy, Applied Mechanics, and Practical Geology, Düsseldorf, 1910.

As already announced in *Bulletin No.* 32, August, 1909, the Congress will meet at Düsseldorf during the last week in June, 1910. Extensive preparations are in progress, including visits to technical institutions and industrial establishments, excursions to localities of geologic interest, etc., which will have a direct bearing on the papers and addresses presented at the meeting.

The following information, dated October, 1909, has been received from the Secretary of the Congress:

Membership:

There will be two grades of membership of the Congress, as follows:

1. Members, who are entitled to become patrons of the Congress by payment to the funds of a contribution of not less than 100 marks (£5).

2. Members, who pay a subscription of 20 marks (£1).

The first-named class of members or patrons will be entitled to receive the printed reports of all proceedings of the Congress and of all its sections. Members of the second class, or ordinary members, will receive the reports of that section only in which they enroll themselves. The proceedings of any one of the other sections are obtainable at an additional charge of 5 marks (5s.) in each case.

Meetings and Excursions:

The work of the Congress will be performed:

1. In General Meetings, at which various papers of general interest will be presented.

2. In sectional Meetings, for the purpose of discussing important problems relating to Mining, Metallurgy, Applied Mechanics, and Practical Geology.

3. By making visits to scientific Institutions and industrial undertakings, etc., and by excursions to districts of geological interest.

For the purpose of entertaining ladies accompanying members to the Congress, a Ladies' Committee will be formed, which will endeavor in every way to render the stay of the lady visitors in Düsseldorf as agreeable and enjoyable as possible.

Provisional Program:

Section I: Mining.

1. Shaft-sinking, with special reference to the cement processes, freezing processes, and tubbing of shafts at great depths. The lining of shafts with concrete and reinforced concrete.
2. Methods of working, mine supports, with special reference to hydraulic packing, the use of reinforced concrete, the preservation of timber, and lighting of collieries.
3. Winding and haulage, with special reference to winding ropes, safety-catches and appliances, underground haulage, and haulage from the working faces.

4. Mine drainage.
5. Risks arising from fire-damp, coal-dust, and underground fires, and the methods of combating these.
6. The mechanical preparation of coal and ore, the recovery of bye-products, briquetting, and the utilization of low-grade fuels.
7. Recent practice in mine surveying.
8. Statistics.
9. Sanitation and hygiene.

Section II: Metallurgy.

A. Production of pig iron.

1. Coking.
   (a) Ovens.
   (b) Mechanical appliances.
   (c) Recovery of bye-products.
2. Ore supply.
   (a) Recent discoveries of ore.
   (b) Recent development and prospects of the ore briquetting processes.
3. Metallurgy of the blast-furnace process.
   (a) Influence of foreign substances.
   (b) Composition of slag.
4. Blast-furnace working.
   (a) Ore conveyance, storage, and charging.
   (b) Gas-washing and purification of the waste water.
   (c) Dry air-blast.
   (d) Casting machines and mixers.
5. Utilization of waste products.
   (a) Gases.
   (b) Dust from blast-furnace gases.
   (c) Slag (for hydraulic packing, cement, stone, concrete).

B. Production of malleable iron.

1. Advances in the methods of the metallurgical treatment of iron and steel.
   (a) Air-blast refining processes.
   (b) Open-hearth refining processes.
   (c) Processes for the production of electro-steel.
2. Production and treatment of special alloys of steel.

C. Iron and steel manufacture.

1. Improvements in the methods of casting iron and steel.
2. Further treatment of malleable iron.
   (a) Forging and pressing.
   (b) Rolling.
   (c) Fitting.
   (d) Development of the welding processes.
3. The driving of rolling-mills considered technically and economically (steam, gas, electricity).

D. Testing of iron and other metals.

1. Chemical testing.
2. Mechanical testing.

3. Metallography and microscopy of metals.

E. Economics of the iron industry.

1. Iron trade statistics.
2. Labor conditions and labor supply.
3. Patent rights.

F. Advances in the metallurgy of non-ferrous metals.

Section III: Applied Mechanics.

1. History of machine construction for mining and metallurgical purposes.
2. Steam raising.
3. Central electric power stations.
   (a) Reciprocating engines (steam, gas).
   (b) Turbo-engines.
4. Central condensing plant.
5. Winding engines.
   (a) Steam winders.
   (b) Electric winders.
   (c) Safety and signalling apparatus.
6. Pumping.
7. Fans and air-compressors.
8. Blowing engines for blast-furnaces and steel works.
   (a) Reciprocating blowers.
   (b) Turbo-blowers.
9. Methods of driving rolling-mills.
10. Rolling-mills and accessories.
11. Conveyors for mining and smelting works.
   (a) For materials in bulk (coal or coke).
   (b) Special cranes and ladle-cars.
   (c) Loading and unloading apparatus.

Section IV: Practical Geology.

1. Importance of practical geology in science and political economy.
2. Stratigraphy and genesis of the available mineral deposits. Calculations of their yearly output and resources.
3. Seismology, terrestrial magnetism, and terrestrial heat.
4. Questions relating to hydrology.
5. The utilization of natural sources of water power. Barrages.

The Committee on Organization request that applications for membership, accompanied by a remittance through the Post Office of the amount of the subscription, be made as soon as possible, in any case not later than Mar. 1, 1910. The remittance should be made payable to the "Stahlwerksverband A.-G. Düsseldorf, Kongress 1910."

Inquiries concerning membership in the Congress, announcement of proposed papers, length and time of presentation, suggestions, etc., may be addressed to Dr. E. Schrödter, Secretary of the Committee of the Congress, Düsseldorf, Jacobistrasse 3/5, Prussia.

Office Facilities for Visiting Members.

A separate room in the suite occupied by the American Institute of Mining Engineers on the ninth floor of the United Engineering Society Building, has been equipped with furniture and telephone extension for the temporary use of members of the Institute or of sister societies, or visitors suitably accredited.

Members of the Institute visiting New York for a short time, who need office facilities during their stay, or members residing in the city who need temporary office accommodation, can arrange to have set apart for their exclusive use a room, equipped with office furniture, telephone, etc., in the suite of the Institute. It is not the intention to give possession of the room to any individual for an indefinite time, but to offer to members of the Institute an opportunity to acquire a well-located, well-equipped business headquarters to carry on transactions which would not warrant the establishment of a permanent office. The room devoted to this purpose is entirely separate from the reception- and writing-rooms for the general use of the members. A small fee will be required for the use of the facilities furnished. For the conditions of this privilege, inquiry should be made at the office of the Secretary of the Institute.

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In short, to those who own complete sets of the *Transactions*, this Index will be a great convenience; but to those who do not, it will be a professional necessity.

This volume is an octavo of 706 pages, containing more than 60,000 entries, duly classified with sub-headings, and including abundant cross-references. It has not been stereotyped, and the edition is limited to 1,600 copies. The price of the volume, bound in cloth, is $5, and bound in half-morocco to match the *Transactions*, $6. The delivery charges will be paid by the Institute on receipt of the above price.

Hydrographic Chart.

Owing to the great value to hydrographers of the chart contained in the paper, A Graphic Solution of Kutter's Formula, by L. I. Hewes and Joseph W. Roe (*Bulletin No.* 29, May, 1909, p. 454), a special edition for office or field use has been printed on durable cloth. Copies of this separate chart may be obtained, at a cost of 50 cents each, on application to the office of the Secretary of the Institute.

Special Notice.

Attention is respectfully requested to the announcement on pages 10 and 11 of the advertising section of this *Bulletin* concerning certain books and periodicals needed to complete sets in the Library.

Spokane Meeting.

Copies of the "Proceedings of the Spokane Meeting," which is Paper No. 3 of this *Bulletin*, printed in pamphlet form with special covers, may be obtained at a cost of 50 cents each, on application to the office of the Secretary of the Institute.

LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.

The libraries of the above-named Societies are open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August.

RULES.

For the protection and convenience of members, the following rules have been adopted:

The Secretary of each Society will, upon application, issue to any member of his Society in good standing a personal, non-transferable card, entitling him to the use of the Libraries in the alcoves of the Reading-Room.

This card, as well as any card of introduction given to a non-member, must be signed by the person receiving it, and surrendered at the desk at the time of its presentation. At every visit he must identify himself by signing his name in the registry.

Strangers who desire to enjoy the privilege of entering the alcoves are requested to present either letters of introduction from members, or cards, such as will be furnished upon application by the Secretary of each Society. The first two alcoves are free to all; and admission to the inside alcoves is given upon proper introduction.

The above rules apply to all persons except officers of the three Societies, personally known as such to the librarians.

The librarians are not permitted to lend to any person any catalogued pamphlet or volume, unless authorized in writing so to do by the Secretary or Chairman of the Library Committee of the Society to which the pamphlet or volume belongs.

Any person discovering a mutilation or defect in any book of the libraries is requested to report it to the librarian on duty.

Library Additions.

From Nov. 1 to Dec. 1, 1909.

[Copies of the list of additions to the Libraries of the American Society of Mechanical Engineers and the American Institute of Electrical Engineers can be obtained on application to the Secretary of the American Institute of Mining Engineers.]

ADMIRAL PRINDLE MINING COMPANY. Description of. New York, n. d. (Purchase.)

AGUACATE MINES, INC. Description of. New York, n. d. (Purchase.)

ALASKA GOLD STANDARD MINING COMPANY. Sinking Fund Gold Bonds, Jan. 21, 1901. N. p., n. d. (Purchase.)

ALASKA TRANSPORTATION AND DEVELOPMENT COMPANY. Salutatory. Chicago, 1897. (Purchase.)

AMERICAN INSTITUTE OF MINING ENGINEERS. Yukon Territory, Canada. (Souvenir of Visit, July, 1905.) (Purchase.)

AMERICAN MINES COMPANY. Description of Weaver Mining District, Yavapai County, Arizona. N. p., n. d. (Purchase.)

AMERICAN MINING CODE. By H. N. Copp. Ed. 2. Washington, 1886. (Purchase.)

DEM ANDENKEN DES GEHEIMEN BERGRATS PROF. DR. HERMANN WEDDING. Berlin, 1908. (Exchange.)

ANGLO-AMERICAN CUBAN COMPANY. Lands and Fortunes in Cuba and Porto Rico. Boston, n. d. (Purchase.)

APPLICATIONS OF ELECTRICITY TO PROPULSION OF NAVAL VESSELS. By W. L. R. Emmet. (Advance paper, Society of Naval Architects and Marine Engineers, Nov., 1909.) (Gift.)

ARIZONA—EXECUTIVE DEPARTMENT. Report of the Acting Governor, 1891. Washington, 1891. (Purchase.)

ARIZONA PROSPECTING, MINING, AND DEVELOPMENT COMPANY OF PHŒNIX. Prospectus, 1899. N. p., n. d. (Purchase.)

ASTURIAN HÆMATITE IRON-ORE SYNDICATE, LTD. Prospectus. London, 1896. (Purchase.)

ATLIN GOLD FIELDS. A Short Account of the District and its Resources. N. p., n. d. (Purchase.)

BALTIMORE COPPER AND GOLD MINING COMPANY. Prospectus, 1900. Toronto, 1900. (Purchase.)

BALTIMORE GOLD MINING AND DEVELOPMENT COMPANY. Prospectus. N. d. (Purchase.)

BATTLE BRANCH GOLD MINING COMPANY. Prospectus. Boston, n. d. (Purchase.)

BATTLE MOUNTAIN CONSOLIDATED GOLD MINING COMPANY. Abridged Prospectus, Oct., 1896. Victor, n. d. (Purchase.)

BATTLE MOUNTAIN CONSOLIDATED GOLD MINING COMPANY. Description of. N. p., n. d. (Purchase.)

BAXTER MINING COMPANY. Zinc and Lead in the Quapaw Reserve. Baxter Springs, n. d. (Purchase.)

BELLE ISLE MINING COMPANY. Prospectus. N. p., n. d. (Purchase.)

BELL-WETHER GOLD MINING COMPANY. On the Mother Lode, Amador County, Cal. Chicago, 1896. (Purchase.)

BERGMANNISCHES LASCHENBUCH, 1791. By A. W. Kohler. Freyberg und Annaberg, 1791. (Purchase.)

BESCHREIBUNG EINES MIT DEM MARKSCHEIDERGONIOMETER AUSGESÜHRTEN WÄHRZUGES. By — Junge. N. p., n. d. (Purchase.)

BLACK BEAR MINING COMPANY. Description. N. p., n. d. (Purchase.)

BLACK HILLS PORCELAIN CLAY AND MARBLE MINING COMPANY. Prospectus. Detroit, n. d. (Purchase.)

BLACK PRINCE COPPER COMPANY. Prospectus. Denver, n. d. (Purchase.)

BOARD OF SUPERVISING ENGINEERS CHICAGO TRACTION. Annual Report, 1st. Chicago, 1908. (Gift of B. J. Arnold.)

BORTLE COPPER-GOLD COMPANY. Prospectus in Brief and Engineer's Report. N. p., 1906. (Purchase.)

BOSTON-WYOMING SMELTER, POWER, AND LIGHT COMPANY. Announcement. N. p., n. d.

BRICKLAYING SYSTEM. By F. B. Gilbreth. New York–Chicago, Myron C. Clark Publishing Co., 1909. (Gift of Author.)

[SECRETARY'S NOTE.—Many of the handicrafts (and especially, perhaps, that of the bricklayers) have been injured by the mistaken policy of the labor-unions in limiting the employment and training of apprentices, and opposing the operation of the trade-schools. This, I hope and believe, is but a temporary incident in the history of our unions. Sooner or later they must certainly realize that the increase of individual skill and efficiency is one of their proper functions. Otherwise, they will only continue a hopeless battle against the irresistible advance of science and practice.

In this book Mr. Gilbreth has undertaken to state that knowledge which has been handed down from journeyman to apprentice for generations; to describe methods which will reduce costs, and at the same time increase the pay of the skilled workman, and to promote by intelligent guidance the progress of apprentices, so that they may become proficient in the shortest possible time. These purposes ought to be—and surely some day will be—approved by working bricklayers. Meanwhile, Mr. Gilbreth's exposition of them, comprising the discussion of the training of apprentices, methods of management and construction, the transportation of material, the building of high chimneys, the manufacture and use of mortar, bricks, brick-laying, brick-layers' tools, etc., will be found most valuable to students, beginners, and superintendents in that line of work.—R. W. R.]

BRITISH COLUMBIA. Information for Prospectors, Miners, and Intending Settlers, 1897. (Purchase.)

BRITISH FOUNDRYMEN'S ASSOCIATION. Proceedings Containing the Report of the Fifth Annual Convention at Newcastle-on-Tyne, Aug. 4–6, 1908. N. p., n. d. (Exchange.)

BRITISH STANDARD SCREW THREADS AND PIPE THREADS. (Publication No. 34, Engineering Standards Committee.) London, 1908. (Gift.)

BRITISH STANDARD SPECIFICATION AND SECTIONS OF STEEL FISH-PLATES FOR BRITISH STANDARD BULL HEAD RAILWAY RAILS. (Publication No. 47, Engineering Standards Committee.) London, 1909. (Gift)

BRITISH STANDARD SPECIFICATION FOR AMMETERS AND VOLTMETERS. (Publication No. 49, Engineering Standards Committee.) London, 1909. (Gift.)

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BRUCE MINES AND ALGOMA RAILWAY. Description. N. d. (Purchase.)

BUCK MOUNTAIN, NEVADA, GOLD MINES COMPANY. Prospectus. Nevada, n. d. (Purchase.)

BUCKEYE MINING COMPANY'S PROPOSITION OF NEVADA CITY, CAL. N. p., n. d. (Purchase.)

——— Report of Stockholders' Meeting, 1903. N. p., n. d. (Purchase.)

BUILDING AND EQUIPPING THE NON-MAGNETIC AUXILIARY YACHT CARNEGIE WITH PRODUCER GAS PROPELLING EQUIPMENT. By W. Downey. (Advance paper, Society of Naval Architects and Marine Engineers, Nov., 1909.) (Gift.)

BULLETIN OF REVENUES AND EXPENSES OF STEAM ROADS IN THE UNITED STATES. Nos. 1–4. Washington, U. S. Government, 1909. (Exchange.)

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CAPE OF GOOD HOPE GEOLOGICAL COMMISSION. Annual Report, 13th. Cape Town, 1909. (Exchange.)

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CARTE GEOLOGIQUE DE L'AMERIQUE DU NORD, 1906. (Gift of Robert T. Hill.)

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CARTERS GOLD MINING COMPANY. Extract from General Reports. San Francisco, n. d. (Purchase.)

CASA DIABLO GOLD MINING COMPANY. President's Annual Report, 1906. Chicago, 1906. (Purchase.)

CATALOGUE OF PUBLICATIONS OF THE GEOLÓGICAL SURVEY OF CANADA. (Revised to Jan. 1, 1909.) Ottawa, 1909. (Exchange.)

CAUCA VALLEY CONSOLIDATED MINES COMPANY. Report and Statement of the Fourth Annual Meeting, June 4, 1906. Detroit, 1906. (Purchase.)

CENTRAL TUNNEL. Detailed Report of Eleven Months' Work. Denver, n. d. (Purchase.)

CHARTERED COMPANY OF LOWER CALIFORNIA. Official Prospectus. Boston, n. d. (Purchase.)

CHEMISTS' CLUB. List of Members, 1909. New York, 1909. (Gift of Club.)

CHEWELAH. Her Mineral Resources and Possibilities. N. p., 1905. (Purchase.)

CHINESE CLASSICS. Part I.—Confucius. By James Legge. New York, 1870. (Purchase.)

CHOPACA GOLD MINING AND MILLING COMPANY. Prospectus. N. p., n. d. (Purchase.)

CLARENDON MINING COMPANY OF ONTARIO, LTD. Description. Windsor, n. d. (Purchase.)

COAL FIELDS OF MANITOBA, SASKATCHEWAN, ALBERTA, AND EASTERN BRITISH COLUMBIA. By D. B. Dowling. Ottawa, 1909. (Exchange.)

COBALT CENTRAL MINES COMPANY. Prospectus. Cobalt, 1908. (Purchase.)

COBALT-IDAHO COMPANY. Description. Detroit, n. d. (Purchase.)

COBALT NEWS. July, 1906. (Purchase.)

COBALT NUGGETS. Feb. 25, 1908. (Purchase.)

COCHISE COPPER MINING COMPANY, LTD. Prospectus. Bisbee, n. d. (Purchase.)

COLONIAL COPPER COMPANY. Report, ed. 5, Jan., 1900. New York, 1900. (Purchase.)

COLORADO GEOLOGICAL SURVEY. Report, 1st, 1908. Denver, 1909. (Gift.)

COLUMBINE-VICTOR DEEP MINING AND TUNNEL COMPANY. Abridged Prospectus, Feb., 1896. Colorado Springs, 1896. (Purchase.)

COMMERCIAL CLUB, CHERIDAN, WYO. Why Sheridan? N. p., n. d. (Purchase.)

COMMERCIAL DEDUCTIONS FROM COMPARISONS OF GASOLINE AND ALCOHOL TESTS ON INTERNAL-COMBUSTION ENGINES. (Bulletin No. 392, U. S. Geological Survey.) By R. M. Strong. Washington, U. S. Government, 1909. (Exchange.)

COMPARATIVE TESTS OF RUN-OF-MINE AND BRIQUETTED COAL ON THE TORPEDO BOAT BIDDLE. (Bulletin No. 403, U. S. Geological Survey.) By W. T. Ray and Henry Kreisinger. Washington, U. S. Government, 1909. (Exchange.)

CONGRÈS GÉOLOGIQUE INTERNATIONAL. Washington, 1891. Procès-Verbaux des Séances, Aug. 26 à Sept. 1, 1891. Washington, 1892. (Purchase.)

CONNERS TIN PRODUCTION COMPANY. Prospectus. Deadwood, n. d. (Purchase.)

COPPER CROWN MINING COMPANY. America's Most Famous Copper Field. St. Louis, n. d. (Purchase.)

CRIPPLE CREEK. Some Facts About. Sept., 1895. N. p., n. d. (Purchase.)

CYCLISTS' ROAD MAP OF NEW YORK. Sections 1–6. (Gift of G. F. Kunz.)

DEFECTS AND DISADVANTAGES OF THE PRESENT FORM OF TRIPODS FOR FIELD-INSTRUMENTS. Philadelphia, 1878. (Purchase.)

DESIGN OF SUBMARINES. By M. F. Hay. (Advance paper, Society of Naval Architects and Marine Engineers, Nov., 1909.) (Gift.)

DETROIT ALASKA MINING COMPANY. Statement. Detroit, 1905. (Purchase.)

DETROIT MICA MINING COMPANY. Prospectus. Detroit, n. d. (Purchase.)

DETROIT TAVICHE MINING COMPANY. Annual Report, 1905. Ocotlan, 1905. (Purchase.)

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DEVELOPMENTS OF COPPER MINES. N. p., n. d. (Purchase.)

DEVELOPMENT OF THE GASOLINE POWER BOAT. By E. T. Keyser. (Advance paper, Society of Naval Architects and Marine Engineers, Nov., 1909.) (Gift.)

DEVONIAN FAUNA OF THE OURAY LIMESTONE. ((Bulletin No. 391, U. S. Geological Survey.) By E. M. Kindle. Washington, U. S. Government, 1909. (Exchange.)

DICTIONARY OF ARTS, MANUFACTURES, AND MINES. Vol. 1. By A. Ure. New York, 1853. (Purchase.)

DINGLER'S POLYTECHNISCHES JOURNAL. Vols. 7–10. Augsburg, 1873. (Purchase.)

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ECLECTIC ENGINEERING MAGAZINE. Vol. I., 1869. New York, Van Nostrand, 1869. (Purchase.)

EDWARD MINING AND MILLING COMPANY. Prospectus. Spokane, n. d. (Purchase.)

EFFECT OF PARALLEL MIDDLE BODY UPON RESISTANCE. By D. W. Taylor. (Advance paper, Society of Naval Architects and Marine Engineers, Nov., 1909.) (Gift.)

ELEMENTOS DE GEOGRAFIA DE LA ISLA DE CUBA. By P. J. M. de la Concepcion. Habana, 1892. (Purchase.)

ELY CENTRAL COPPER COMPANY. Report on the Property in Robinson Mining District, Nev. By W. A. Farish. New York, n. d. (Gift of B. H. Scheftels & Company.)

EMERSON MINING COMPANY. A Few Facts About the Ainsworth Mining Camp. Minneapolis, n. d. (Purchase.)

EMPRESS MINING AND MILLING COMPANY, LTD. Description. Chicago, n. d. (Purchase.)

ESQUISSE GÉOLOGIQUE ET MINÉRALOGIQUE DU DISTRICT AURIFERE DE SANTA-CRUZ, HONDURAS. By A. J. Bourdariat. Bruxelles, 1893. (Reprint, *Bulletin de la Société Belge de Géologie.*) (Purchase.)

EVERGREEN GOLD AND COPPER MINES COMPANY. Description of Company. Apex, Colo., n. d. (Purchase.)

EVOLUTION OF SCREW PROPULSION IN THE UNITED STATES. By C. H. Cramp. (Advance paper, Society of Naval Architects and Marine Engineers, Nov., 1909.) (Gift.)

EXPENDITURE OF PUBLIC MONEY. Editorial from the San Francisco *Chronicle*, Monday, Aug. 9, 1909. (Gift of Civic League of San Francisco.)

FEDERAL GOLD MINING COMPANY OF WYOMING. Examination of the Property and Report. Chicago, n. d. (Purchase.)

FINANCE'S PRIMER. By M. P. Boss. San Francisco, 1895. (Purchase.)

FLORIDA STATE GEOLOGICAL SURVEY. Annual Report, 1st, 1907-08. Tallahasse, 1908. (Exchangee.)

——— Bulletin No. 1. Tallahassee, 1908. (Exchange.)

FOREIGN TRADE MERCHANT MARINE OF THE UNITED STATES: CAN IT BE REVIVED? By G. W. Dickie. (Advance paper, Society of Naval Architects and Marine Engineers, Nov., 1909.) (Gift.)

FOREIGNERS AND FOREIGN CORPORATIONS IN MEXICO. Their Rights and Obligations Under the Law (Book A). Mexico, n. d. (Gift of Mexican Title Mortgage Co.)

FOSS RIVER CONSOLIDATED COPPER COMPANY. General Manager's Report to the Directors. Everett, 1902. (Purchase.)

FRASER AND CHALMERS. Catalogues. Parts 1–2. Chicago-London, n. d. (Purchase.)

GEHEIMER BERGRAT PROFESSOR DR. HERMANN WEDDING. (Reprint, *Stahl und Eisen*, No. 21, 1908.) (Exchange.)

GEM COPPER MINING COMPANY, LTD. Prospectus. Priest River, Idaho, n. d. (Purchase.)

GEOLOGY AND UNDERGROUND WATERS OF SOUTH DAKOTA. (Water-Supply Paper No. 227, U. S. Geological Survey.) By N. H. Darton. Washington, U. S. Government, 1909. (Exchange.)

GEOLOGY OF ORE-DEPOSITS. By H. H. Thomas and D. A. MacAlister. London, E. Arnold, 1909. (Gift of Longmans, Green & Co.) Price, $2.50.

[SECRETARY'S NOTE.—This handy 12mo volume of 416 pages belongs to Arnold's Geological Series, of which Dr. J. E. Marr, F.R.S., is the General Editor, and in which the treatise of Dr. Walcot Gibson on the geology of coal and coal-mining has already appeared. Dr. Marr, in his preface, vouches for the practical acquaintance of the authors with their subject and its literature; and the authors, in their preface, apologize for the shortcomings inevitably involved in the attempt to condense into small space the results of their observation and study. In view of these obvious limitations, they have done their work very well, producing a useful summary of the science of ore-deposits as it stands to-day. The book treats successively of ores due to the differentiation of igneous magmas, pneumatolysis, hydatogenesis, metasomatic replacement, precipitation, metamorphosis, and other secondary processes, and detrital and alluvial deposition. It ought to reach a second edition; and if the authors will then supplement it with a critical bibliography and references to authorities (which, I think, could be done without serious increase of size) it would be not only an intelligent summary, but an excellent guide to detailed further study.—R. W. R.]

GEORGIA RAILROAD LAND AND COLONIZATION COMPANY. Letter Regarding Removal. Augusta, 1897. (Purchase.)

GIVEN'S PAPER ABOUT GOLDFIELDS OF NEVADA. Vol. 2, No. 4. N. p., n. d. (Purchase.)

GLADYS A MINING COMPANY. Prospectus. Cripple Creek, n. d. (Purchase.)

GOLD BUG MINING COMPANY, LTD. Prospectus. N. p., n. d. (Purchase.)

GOLD COIN MINING AND LEASING COMPANY. Prospectus. Colorado Springs, n. d. (Purchase.)

GOLD COIN MINING AND MILLING COMPANY. Prospectus, 1904. Detroit, 1904. (Purchase.)

GOLD DUST FROM ROCKY BAR. By E. L. Proctor. Lynn, n. d. (Purchase.)

GOLD EXTRACTION MINING & SUPPLY COMPANY. Report on the Properties of in Sugar Loaf, Gold Hill, and Ward Mining Districts. Denver, n. d. (Purchase.)

GOLD PAN MINING COMPANY. Brief History, Nov., 1902. Breckenridge, n. d. (Purchase.)

GOLD FIELDS OF THE EAST AND THEIR LOW-GRADE ORES. Address, Sept. 12, 1903. By C. L. Dignowity. Deadwood, 1903. (Purchase.)

GOLD REGIONS OF THE STRAIT OF MAGELLAN AND TIERRA DEL FUEGO. By R. A. F. Penrose, Jr. (Reprinted from the *Journal of Geology*, Vol. XVI., No. 8, Nov.-Dec., 1908.) (Gift of Author.)

GOLD RESOURCES OF COLORADO. Denver, n. d. (Purchase.)

GOLD AND SILVER MINING IN MONTANA. By W. T. Mendenhall. Helena, n. d. (Purchase.)

GOLDEN COCHITI TUNNEL COMPANY. Prospectus, 1900. Albuquerque, N. M., 1900. (Purchase.)

GOLDEN OPPORTUNITIES IN MINING AND TOWN LOT INVESTMENTS AT PEARL, COLORADO. By A. N. Hosking. N. p., n. d. (Purchase.)

THE GRAHAM PLACER. By J. Houghton. Detroit, 1897. (Purchase.)

GO SOUTH, YOUNG MAN. Proposal for the Establishment of a South American Exploration, Development, and Colonization Company. By F. W. Grauert. N. p., n. d. (Purchase.)

GRANITE MOUNTAIN MINING COMPANY. Annual Report, 1891. Montana, 1891. (Purchase.)

GRANITES OF VERMONT. (Bulletin No. 404, U. S. Geological Survey.) By T. N. Dale. Washington, U. S. Government, 1909. (Exchange.)

GREAT BRITAIN—MINES AND QUARRIES. General Report and Statistics for 1908. Part III.—Output. London, 1909. (Exchange.)

GREAT KABORA MINING COMPANY. Report of First Meeting of Trustees, 1904. Washington, 1904. (Purchase.)

GREAT WESTERN MINING COMPANY. Description. Kingman, n. d. (Purchase.)

GREATER AMERICAN MINING COMPANY. Nome District Report, 1902. New York, 1902. (Purchase.)

——— Prospectus, the Gold Fields of Nome, Alaska. New York, n. d. (Purchase.)

GREATER ONTARIO TO BE DEVELOPED BY THE BRUCE MINES AND ALGOMA RAILWAY, Which is the Western Ontario Route to Hudson Bay. N. p., n. d. (Purchase.)

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GROUSE MOUNTAIN MINING AND MILLING COMPANY. Prospectus of. Colorado Springs, n. d. (Purchase.)

GRUNDLAGEN DES RECHNUNGSWESENS. By C. G. Gottschalk. Leipzig, 1865. (Purchase.)

HAMMOND GOLD REEF MINING COMPANY, LTD. Prospectus. Toronto, n. d. (Purchase.)

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HATTIE BELLE GOLD, COPPER, AND NICKEL COMPANY. Prospectus. Sunderland, Ont., n. d. (Purchase.)

HEPBURN GOLD MINING COMPANY. Statement. San Francisco, n. d. (Purchase.)

HERCULES GOLD MINING COMPANY. Prospectus. Chicago, n. d. (Purchase.)

HISTORY OF THE CLAY-WORKING INDUSTRY IN THE UNITED STATES. By Heinrich Ries and Henry Leighton. New York, J. Wiley & Sons, 1909. Price, $2.50 net. (Gift of Publishers.)

[SECRETARY'S NOTE.—This volume is one of the series prepared under the auspices of the Carnegie Institution in connection with a projected Economic History

of the United States, and its separate publication in advance, like that of other similar volumes, has been wisely permitted by that institution. If I understand this action correctly, it means that the books thus printed beforehand will be treated hereafter as materials for the projected history, and not simply reprinted as parts of it. And I have called this policy wise, because some of the volumes which have already appeared in accordance with it are lamentably crude and inadequate, and need to be both closely sifted and greatly reinforced in the preparation of a comprehensive work. But this particular volume, it seems to me, might well be adopted as it stands. Professor Ries has already won recognition as an authority on the occurrence, nature, and industrial treatment of the clays of the United States, and this work evinces throughout the conscientious care with which the history of the subject has been studied and stated. Part I. presents a review of the industry, classified according to its products and their uses, and Part II. gives a historical account of it by States. As a whole, the book will be a revelation to many of the extent and importance of a branch of mining which surpasses in the value of its products many other branches more romantically esteemed and more sensationally advertised.—R. W. R.]

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INCIDENTAL PROBLEMS IN GAS-PRODUCER TESTS. (Bulletin No. 393, U. S. Geological Survey.) By R. H. Fernald and others. Washington, U. S. Government, 1909. (Exchange.)

INDIAN RIVER GOLD MINING COMPANY. Prospectus. Boston, n. d. (Purchase.)

INFLUENCE OF THE POSITION OF THE MIDSHIP SECTION UPON THE RESISTANCE OF SOME FORMS OF VESSELS. By H. C. Saddler. (Advance paper, Society of Naval Architects and Marine Engineers, Nov., 1909.) (Gift.)

INFORME RENDIDO AL CONSEJO DE ADMINISTRACIÓN DE LA CIA. MINERA "ELDORADO." By D. J. Flebbe. Mexico, 1908. (Purchase.)

INTERNATIONAL MINING AND MILLING COMPANY. Mr. Bouleau's Report. Detroit, 1906. (Purchase.)

INTER-STATE LAND BUREAU. Solid Facts about the Gold Fields of the Southern States. Washington, 1894. (Purchase.)

IRON-ORE DEPOSITS IN SYDVARANGER AND RELATIVE GEOLOGICAL PROBLEMS. N. p., 1902. (Purchase.)

ISTHMIAN CANAL. By H. H. Rousseau. Washington, U. S. Government, 1909. (Exchange.)

ISTHMIAN CANAL COMMISSION. Annual Report, 1909. Washington, U. S. Government, 1909. (Exchange.)

JAHRBUCH FUR DEN BERG-UND HUTTEN MANN, 1866. Freiberg, 1866. (Purchase.)

JALISCO COPPER MINING COMPANY. Description. Nogales, Ariz., n. d. (Purchase.)

JOE WONDER MINING COMPANY. Location of Claims. Wonder, Nev., n. d. (Purchase.)

KENTUCKY COAL, COKE, AND TIMBER. Illustrated Description of the Log Mountain Clear Creek Region. Frankfort, 1890. (Purchase.)

KENTUCKY—GEOLOGICAL SURVEY. Reports on Eastern Coal Field. Frankfort, 1884. (Purchase.)

KINTYRE MINING AND SMELTING COMPANY. Prospectus. Rossland, B. C., n. d. (Purchase.)

KLONDYKE AND STEWART RIVER MINING COMPANY. Prospectus. Chicago, n. d. (Purchase.)

KNICKERBOCKER GOLD MINING, MILLING, TUNNEL, AND TRANSPORTATION COMPANY. Description of Property. Colorado, n. d. (Purchase.)

KNICKERBOCKER SYNDICATE PRESENTS THE 6 PER CENT. BONDS OF THE COLORADO YULE MARBLE COMPANY TO MR. W. M. COURTIS. New York, 1904. (Purchase.)

KOOTENAI CARIBOO MINING AND INVESTMENT COMPANY, LTD. Report. Toronto, n. d. (Purchase.)

LACKAWANNA STEEL COMPANY. Catalogue of Open-Hearth and Bessemer Steel Products, 1909. (Gift of Lackawanna Steel Company.)

LADUE, JOSEPH, GOLD MINING AND DEVELOPMENT COMPANY OF YUKON. Official Prospectus. N. p., n. d. (Purchase.)

LA GRANDE GOLD MINING COMPANY. What the Highest Authorities Say About What We Have. Bloomsburg, Pa., n. d. (Purchase.)

LANDSLIDES IN THE SAN JUAN MOUNTAINS, COLORADO. (Professional Paper No. 67, U. S. Geological Survey.) By Ernest Howe. Washington, U. S. Government, 1909. (Exchange.)

LEAGUE OF AMERICAN WHEELMEN. Bulletin and Good Roads. Vol. 23, Nos. 18–24; Vol. 24, Nos. 10–26; Vol. 25, Nos. 1–18; Vol. 26, Nos. 2–3, 6, 11, 14, 18, 20, 21, 23–27; Vol. 27, Nos. 1–9, 13, 15–18, 20–23; Vol. 28, Nos. 3, 12, 13, 15–16, 18–27; Vol. 29, Nos. 1, 3–7, 15, 26. Boston, 1896–1899. (Gift of George F. Kunz.)

LEDYARD GOLD MINES COMPANY, LTD. Description. Toronto, n. d. (Purchase.)

LEHRE VON DEN ERZLAGERSTÄTTEN. (Science of Mineral Deposits.) By Dr. Richard Beck. Ed. 3, Vols. 1–2. Berlin, 1909. (Gift of Author.)

[SECRETARY'S NOTE.—The appearance of this third edition of Professor Beck's treatise (of which the second edition, issued in 1903, has been exhausted), is at once a proof of the high value attached to the work by instructors, students, and practitioners throughout the world, and an occasion for special gratitude on their part, that the accomplished author has been able to bring up to date his excellent critical summary of what has become almost a new science. Professor Beck occupies at Freiberg a chair made famous by distinguished predecessors, some of whom, through the work of their students, as well as their professional publications, acquired a commanding influence upon the theory and practice of mining. The primacy of the Freiberg school was indeed destroyed through the increase, both in the extent of mining-operations and in the number of technical schools for the education of mining engineers; but it was never surrendered to any other superior authority. Up to a certain time, Freiberg—after that time, nobody—dominated that professional field. Since that time, the influence of Freiberg has been measured by the value of its contributions to technical theory and practice; and, in this respect, Professor Beck has done much to maintain the ancient *prestige* of the Academy. In the great modern development of the science of ore-deposits, he was among the first to recognize the importance of the work of American economic geologists; and, in acknowledgment of that candid recognition, as well as of his other claims to professional reward, the American Institute of Mining Engineers

made him an Honorary Member. This latest edition of his treatise shows abundantly his continued esteem for American work in that field, together with his comprehensive, candid, and systematic summary of the recent technical literature of the subject. One of the important features of this edition is the new scientific classification of ore-deposits adopted by the author, under the pressure of progressive developments in genetic theory, etc. Unfortunately, this significant innovation cannot be here and now discussed. The book contains also many new descriptions of productive mining-districts, etc.—R. W. R.]

LE ROI MINING AND SMELTING COMPANY. Prospectus. Spokane, n. d. (Purchase.)
LIFE AND LETTERS OF JOHN BROWN. By E. B. Sanborn. Boston, 1885. (Purchase.)
LILLIAN MINING COMPANY OF PORTLAND, ME. Description. Portland, n. d., (Purchase.)
LIST OF BOOKS IN THE READING ROOM OF THE JOHN CRERAR LIBRARY, 1909. Chicago, 1909. (Exchange.)
LOS REYES GOLD MINING AND MILLING COMPANY. Description of. Oaxaca, n. d. (Purchase.)
LUMBER, LATH AND SHINGLES, 1908. (U. S. Census Bureau. Foreign Products, No. 2). Washington, U. S. Government, 1909. (Exchange.)
MAGALIA PLACER MINING COMPANY. Description of. Los Angeles, n. d. (Purchase.)
MAMMOTH GOLD MINING COMPANY, LTD. Description of property. N. p., n. d. (Purchase.)
MANCHESTER ASSOCIATION OF ENGINEERS. *Transactions*, 1908–09. Manchester, 1909. (Exchange.)
MAP OF QUEEN MINERAL RANCH, GILPIN COUNTY, CAL., 1886. (Purchase.)
MARIPOSA COUNTY. ITS SCENIC WONDERS, ITS GOLD FIELDS. Mariposa, 1904. (Purchase.)
MASCOTTE GOLD MINING COMPANY OF COLORADO. Prospectus. Alma, n. d. (Purchase.)
MASTER CAR BUILDERS' ASSOCIATION. Report of the Proceedings of the Annual Convention, 43d. Chicago, 1909. (Gift.)
MATERIAL HANDLING EQUIPMENTS FOR LAKE VESSELS. By R. B. Sheridan. (Advance Paper, Society of Naval Architects and Marine Engineers, Nov.., 1909.) (Gift.)
MEMOIR OF DOUGLASS HOUGHTON, FIRST STATE GEOLOGIST OF MICHIGAN. By A. Bradish. Detroit, 1889. (Purchase.)
MERCUR GOLD FIELDS (OUR OWN JOHANNESBURG). N. p., n. d. (Purchase.)
METALLURGICAL REVIEW. Vols. 1–2. New York, 1877–1878. (Purchase.)
METALLURGY OF THE COMMON METALS. Ed. 2. By L. S. Austin. San Francisco, *Mining and Scientific Press*, 1909. (Gift of Publishers.)

[SECRETARY'S NOTE.—Prof. Austin, in his work as Professor of Metallurgy at the Michigan College of Mines, has naturally been obliged to condense and summarize, for purposes of instruction, a vast amount of personal observation and experience, as well as the results of wide reading. It is from such sources that our most valuable professional text-books have come, and every such book, born of the exigencies and problems of the lecture-room, the recitation-room, and the laboratory, is valuable to both instructors and students. At the same time, it usually betrays the special knowledge of its author, by the distinction between its

perfunctory treatment of some subjects and its full, masterly handling of others. This book is an instance. Prof. Austin's brief chapter on the metallurgy of iron (that of steel being omitted altogether) is but a meager outline, barely sufficient to give the beginners a general notion of the elementary principles and the usual apparatus of that branch of metallurgy, whereas, his chapters on lead, copper, and connected departments of metallurgical industry, especially in the West, are notably full, fresh, and suggestive. On these subjects, Prof. Austin is himself an authority of no mean weight; and his book is therefore both less and more than a pedagogical scheme; less, because it does not cover all departments with equal thoroughness and detail; more, because in certain departments it represents fully, and with critical suggestions, the best modern theory and practice. The book is illustrated with 295 pictures of furnaces, machinery, etc., which have been intelligently selected and excellently engraved, and constitute, of themselves, a feature of great value.—R. W. R.]

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KLONDIKE OFFICIAL GUIDE. Canada's Great Gold-Field, the Yukon District. By William Ogilvie. Buffalo, 1898. (Purchase.)

ORE-DRESSING. Vols. 3–4. By R. H. Richards. New York, McGraw-Hill Book Co., 1909. (Gift of Publishers.)

[SECRETARY'S NOTE.—I have heard somewhere that, at a recent meeting of a technical society, in the course of a discussion of ore-dressing, a fearless speaker pronounced the one overwhelming present necessity to be a man who would "concentrate Richards!" My informant added that Prof. Richards himself promised to be that man! If all this is true, it is good news to students and practitioners of ore-dressing. Nobody would be as competent as Prof. Richards to condense and summarize his great work. Yet, on the other hand, no summary, even from his hand, could take the place of the original voluminous treatise among those engineers who are burdened with the responsibility of designing great plants to deal with special problems of concentration. For all the modern problems are special and local; and those who must deal with them cannot be satisfied with the generalized deductions ef even the highest authorities. What they need is the details upon which such conclusious are founded, and the means of studying for themselves the reasoning based thereon. I feel sure, therefore, that no condensation of Prof. Richards's treatise, even though it were made by himself, can supersede in value the larger work, of which the two concluding volumes lie before me.

These volumes are essentially supplements to the two preceding ones, the revision and augmentation of which was practically impossible. They treat, therefore, largely by way of postscript, with the topics of Vols. I. and II., such as the general principles of ore-dressing, rolls, stamps, pulverizers, screens, classifiers, jigs, slime-concentrators, magnetic and other separators, general principles and rules of milling, suggestions, tables, bibliography, etc. In short, they express a creditable attempt to bring the treatment of this subject up to date, and they include invaluable data of practice, the importance and helpfulness of which will never be lost.—R. W. R.]

ORIENTAL CONSOLIDATED MINING COMPANY. Report for year ending June 30, 1909. New York, 1909. (Gift of Oriental Consolidated Mining Company.)

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BAKER, T. Treatise on the Mathematical Theory of the Steam-Engine. London, 1864.

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BOURNE, JOHN. Handbook of the Steam-Engine. New York, 1865.

HAUPT, HERMAN. General Theory of Bridge Construction. New York, 1866.

NASON, H. B. Manual of Qualitative Blowpipe Analysis. Philadelphia, 1881.

NYSTROM, J. W. Technological Education and the Construction of Ships and Screw Propellers for Naval and Marine Engineers. Ed. 2. Philadelphia, 1866.

PERRY, M. C. United States-Japan Exhibition. Vols. 1–3. Washington, 1856.

TURNBULL, JOHN. Short Treatise on the Compound Engine. Glasgow, 1873.

U. S. COAST SURVEY. Coast Pilot of Alaska (part 1), 1869. Washington, 1869.

U. S. NAVY DEPARTMENT. Report of the Secretary, 1867, 1873, 1876, 1880, 1885; Vol. 1, 1887. Washington, 1867, 1873, 1876, 1880, 1885, 1887.

U. S. NAVY DEPARTMENT. Office of Naval Intelligence. Annual, July, 1892. Washington, 1892.

WARD, J. H. Elementary Instruction in Naval Ordnance and Gunnery. New York, 1861.

MEMBERSHIP.

NEW MEMBERS.

The following list comprises the names of those persons elected as members who accepted election during the month of November, 1909:

Members.

Henry T. Beckwith,	Philadelphia, Pa.
Fred H. Bostwick,	Denver, Colo.
Horace C. Dickey,	Columbia, Pa.
William B. Foote,	Geneva, N. Y.
Charles H. Grill,	Bob, Nev.
Cleaveland Hilson,	Electric, Mont.
Richard J. Horschitz,	Haileybury, Ontario, Can.
Robert G. Lassiter,	Virgilina, Va.
Alexander Longwell,	Toronto, Ontario, Can.
James H. Macia,	Tombstone, Ariz.
Albert D. Oberly,	Scottdale, Pa.
Marmaduke Peckitt,	Wharton, N. J.
Jesse C. Porter,	Brooklyn, N. Y.
Michael J. Slattery,	San Martin Hidalgo, Jalisco, Mex.
Donald Steel,	Brownsville, Cal.
William C. Stratton,	Scottdale, Pa.
James B. Torbert,	Jersey Shore, Pa.
William Wearne,	Hibbing, Minn.

Honorary Member.

Alexandre Pourcel,	Paris, France.

CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed for election as members of the Institute during the month of November, 1909. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

Members.

Nelson G. Brayer, ... Sharon, Pa.
Halsted Woodrow Caldwell, ... Thomas, Ala.
Arthur O. Christensen, ... Beaufort, S. C.
Harvey Edward Ewing, ... Worth, W. Va.
Thomas W. Graham, ... Workington, England.
John Frederick Hoffman, ... Mammoth, Cal.
Frank Utt Humbert, ... New York, N. Y.
Frank G. Morris, ... Sayreton, Ala.
William Neilson, ... Midas, Nev.
Philip Benjamin Osborn, ... Johannesburg, South Africa.
William James Penhallegon, ... Birmingham, Ala.
Walter Irving Phillips, ... Chicago, Ill.
Irving Lamphry Ryder, ... San José, Cal.
Hedley V. Smythe, ... York Harbor, Newfoundland.
Karl M. Way, ... Boston, Mass.
Arthur Yates, ... Lebong Soelit, Sumatra.

CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of November, 1909. This list, together with the lists given in the *Bulletin*, Nos. 26 to 35, for February to November, therefore, supplements the annual list of members corrected to Jan. 1, 1909, and brings it up to the date of Dec. 1, 1909. The names of Members who have accepted election during the month (new members), are printed in *italics*.

ADAMS, ARTHUR K., Mineral Inspector.......General Land Office, Phœnix, Ariz.
ADAMS, CUYLER, Prest. and Genl. Mgr., Cuyana & Iron Range Ry. Co., Deerwood, Minn.
ADAMS, HUNTINGTON, Min. Eng., Mgr. Cia Minera de Natividad y Anexas, S. A., Natividad, Ixtlan, Oax., Mexico.
ADAMS, MASON T.......................................412 McPhee Bldg., Denver, Colo.
AMBROSIUS, JULIUS R ....Calle Orizaba 42, Colonia Roma, Mexico City, Mex.
AUSTIN, L. S........ ....251 W. 2d St., Salt Lake City, Utah.
BALL, SYDNEY H...71 Broadway, New York, N. Y.
**Beckwith, Henry T.*, Field Geol.........370 Bullitt Bldg., Philadelphia, Pa. '09.
BELL, DR. ROBERT..............................136 McLaren St., Ottawa, Ont., Canada.
**Bostwick, Fred H.*, Genl. Westn. Mgr., Wellman-Seaver-Morgan Co., 611 Ideal Bldg., Denver, Colo. '09.
BRITT, RICHARD H......Care E. Meyer, Jr., & Co., 7 Wall St., New York, N. Y.
BROOKS, RAYMOND...P. O. Box 863. Goldfield, Nev.
BROWN, HARVEY S................ ....Daly West Mine, Park City, Utah.
BROWNE, ARTHUR B.....................................1572 Fillmore St., Denver, Colo.
CHANEY, R. GORDON, JR.......... ....Tonopah, Nev.
COOTE, CHARLES E., Min. Engr., Mount Lyell Comstock Copper Co., Ltd., North Lyell, Tasmania.

CRIST, FREDERICK G., Care Reardon & Crist Cons. Co.,
1166 Webster St., Oakland, Cal.
CRUM, J. RICHMOND.................Great Fenton House, Stoke-on-Trent, England.
DAVELER, ERLE V., Min. Engr., Utah Copper Co.....................Garfield, Utah.
DE ARMOND, CHARLES F., Crown Mines, Ltd., W. Fordsburg, Transvaal, So. A.
DEWEY, WILLIAM P..........................2662 Vermont Ave., Los Angeles, Cal.
*Dickey, Horace C., Mech. Engr.......................................Columbia, Pa. '09.
EAGAN, CHARLES E., Min. Engr.................................Haileybury, Ont., Can.
FLYNN, FRANCIS N., Met.....................Arizona Copper Co., Ltd., Clifton, Ariz.
*Foote, William B., Min. Engr., Blast Furn. Supt.,
17 Genessee St, Geneva, N. Y. '09
FOX, ARTHUR C..............................Washington and Olive Aves., Pomona, Cal.
GEORGE, JOHN G. G..........................Nacozari Hotel, Nacozari, Son., Mexico.
GMELIN, ERNEST, Cons. Min. Engr...........Apartado 57, Aguascalientes, Mexico.
GRIDER, RICHARD L., Min. Engr........................268 E. 14th St., Eugene, Ore.
*Grill, Charles H., Mine Supt.........Lodi Mines Co., Bob, via Luning, Nev. '09.
HALEY, D. F...320 Security Bldg., St. Louis, Mo.
HAVARD, FRANCIS T., Asst. Prof. of Min. and Met.,
Univ. of Wisconsin, Madison, Wis.
HECK, ELMER C...................Care Shannon Engineers, Box 1187, Metcalf, Ariz.
HEISE, A. ROY, Min. Engr. Supt., Port San Luis Refinery, Union Oil Co.,
Port San Luis, Cal.
HOLDEN, EDWIN C....................................1920 Arlington Pl., Madison, Wis.
HOLT, M. B..1420 Josephine St., Denver, Colo.
*Horschitz, Richard J., Min. Engr., P. O. Box 453, Haileybury, Ont., Can. '09.
HUGHES, WILSON W..... ....Apartado 164, Oaxaca, Mexico.
IONIDES, STEPHEN A., Min. and Met. Engr...........................Georgetown, Colo.
IRWIN, DAVID D...Morenci, Ariz.
JENKS, ARTHUR W...................................3909 E. Howell St., Seattle, Wash.
KING, AUSTIN J...Pocahontas, Va.
KITSON, HOWARD W., Boston Cons. Min. Co., Hotel No. 2,
Bingham Canyon, Utah.
KOPELOWITZ, BERTHOLD, Tech. Adviser, General Mining & Finance Corp., Ltd.,
P. O. Box 1242, Johannesburg, Transvaal, So. Africa.
KOZMINSKY, LESLIE M., Boston Cons. Min. Co., Hotel No. 2,
Bingham Canyon, Utah.
LANGTON, JOHN..31 Nassau St., New York, N. Y.
*Lassiter, Robert G., Mine Mgr.......................................Virgilina, Va. '09.
LEER, I. WAYNE VON...Instructed to hold mail.
LINCK, FREDERICK W..................5 Victoria Pl., Eastbourne, Sussex, England.
*Longwell, Alexander, Min. Engr......37 Melinda St., Toronto, Ont., Canada. '09.
LYSER, CHARLES J.......................................Lime Mt., via Tuscarora, Nev.
McCAFFERY, RICHARD S...Moscow, Idaho.
McCANN, FERDINAND..............................American Club, Mexico City, Mexico.
McCARTHY, EDWARD T........ 10 and 11 Austin Friars, London, E. C., England.
MACDONALD, JESSE J..................................427 Kittridge Bldg., Denver, Colo.
*Macia, James H., Supt., Tombstone Cons. M. & M. Co., Ltd.,
Box 555, Tombstone, Ariz. '09.
METCALFE, GEORGE W..................Mammoth Copper Mining Co., Kennett, Cal.
MILLER, JESSE W., Min. Engr.................................Chinacates, Dur., Mexico.
MOTTER, WILLIAM D. B., JR.......Care M. A. Hanna & Co., Crystal Falls, Mich.
MURRAY, WILLIAM F., Asst. Supt.....Victor American Fuel Co., Hastings, Colo.

NAETHING, FOSTER..........................Care Utah Apex, Bingham Canyon, Utah.
NISSEN, PETER N..........................Care E. J. Holland, Cobalt, Ont., Canada.
**Oberly, Albert D.*, Property Engr......H. C. Frick Coke Co., Scottdale, Pa. '09.
OLDFIELD, FRANK W., Min. Engr.............376 Wilcox Bldg., Los Angeles, Cal.
PALMER, EDWARD V.................................1244 Clayton St., Denver, Colo.
**Peckitt, Marmaduke*, Supt. of Mines.................R. F. D., Wharton, N. J. '09.
PHILLIPS, WILLIAM B., Dir. Bureau of Econ. Geol.,
Univ. of Texas, Austin, Texas.
PIDDINGTON, F. L...Cadia, N. S. W., Australia.
**Porter, Jesse C.*, Min. Engr,, 1005 E. 17th St., Flatbush, Brooklyn, N. Y. '09.
PYNE, FRANCIS R...324 Union Ave., Elizabeth, N. J.
REICHARD, MAX...Bergwerk Kappel Littenweiler, b. Freiburg 1, Br., Germany.
RICKARD, ARTHUR J......................10 Treyeu Rd., Truro, Cornwall, England.
SAYRE, MORTIMER F., Engr., with Gila Valley Globe & Northern Ry.,
Globe, Ariz.
SHIPMAN, H. A...600 Marion Bldg., Denver, Colo.
SINN, ALFONSO, Bergwerksdirektor der Boratfelder der
"Dellarocca chem. Fabriken, A. G.," Taltal, Chile, So. Amer.
**Slattery, Michael J.*, Mine Owner, Genl. Mgr. Hacienda de San Vincente,
San Martin Hidalgo, Jalisco, Mexico. '09.
SMITH, ALEXANDER H., Care Bond & Smith, Union Bank Chambers,
Toronto, Ont., Canada.
**Steel, Donald*, Min. Engr. and Geol.............................Brownsville, Cal. '09.
STOCKETT, ALFRED W., Mgr., Simmer & Jack Proprietary Mines, Ltd.,
Box 192, Germiston, Transvaal, So. Africa.
**Stratton, William C.*, Min. Engr.....................Kromer Ho., Scottdale, Pa. '09.
STROUT, ERNEST A., Hotel de Genáve, 8a Calle de Liverpool,
Mexico City, Mexico.
STURGES, HAROLD, Supt., Teziutlan Copper Min. & Smeltg. Co.,
Ejutla, Oax., Mexico.
TENNY, EMIL B.................................4216 Shenandoah Ave., St. Louis, Mo.
THORNE, WILLIAM E., Min. Engr., 805 Alaska Commercial Bldg.,
San Francisco, Cal.
**Torbert, James B.*, Min. Engr.......................................Jersey Shore, Pa. '09.
TOWNSEND, HARRY P., Min. Engr., Village Deep, Ltd., P. O. Box 1145,
Johannesburg, Transvaal, So. Africa.
VAIL, RICHARD H., Asst. Editor, *Engineering and Mining Journal*,
505 Pearl St., New York, N. Y.
WAINWRIGHT, J. HOWARD...............................40 Wall St., New York, N. Y.
WALKER, W. LESTER...... ....P. O. Box 1236, Salt Lake City, Utah.
WASHBURNE, C. W........Cia Mex. de Petroleo, "El Aguila," Tampico, Mexico.
**Wearne, William*, Mine Supt........ ....Inland Steel Co., Hibbing, Minn. '09.
WEICHSEL, OSCAR M...................................2177 E. 82d St., Cleveland, Ohio.
WILSON, ALFRED W. G....Mines Branch, Dept. of Mines, Ottawa, Ont., Canada.
WOODBRIDGE, D. E....................................Providence Bldg., Duluth, Minn.
WOODWARD, W. M. H..Caborca, Son., Mexico,
YOUNG, EDWARD L..................... ....50 Church St., New York, N. Y.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

Name.	Last Address on Records, from which Mail has been Returned.
Alexander, George E.,	Sparta, Ore.
Andersen, Carl,	Johnnie, Nev.
Bartoccini, Astolfo,	214 E. 90th St., New York, N. Y.
Bassett, Thomas B.,	Cumpas, Sonora, Mexico.
Batchelder, Joseph F.,	54 1st St., Portland, Ore.
Bellam, Henry L.,	Reno, Nev.
Bouchelle, James F.,	22 Duncan Ave., Jersey City, N. J.
Brown, Frank H.,	Coppermount, Alaska.
Campa, Jose,	Mexico City, Mexico.
Cragoe, A. Spencer,	Vencedora, Mexico.
Derby, Harry S.,	134 Monroe St., Chicago, Ill.
Dickson, George H.,	Lethbridge, Alberta, Canada.
Dougherty, Clarence E.,	41 Wall St., New York, N. Y.
Ekberg, Benjamin P.,	Johannesburg, Transvaal, So. Africa.
Field, Wilfrid B.,	Mexico City, Mexico.
Francis, George G.,	177 St. George's Sq., London, W., England.
Gage, Edward C.,	San Dimas, Dur., Mexico.
Gee, Emerson,	Reno, Nev.
Hawkins, Tancred,	Ballydehob, Ireland.
Hunt, Thatcher R.,	Iron Mt., via Keswick, Cal.
Jewett, Eliot C.,	2918 Morgan St., St. Louis, Mo.
Kow, Tong Sing,	Shanghai, China.
Mildon, Reginald B.,	Nacozari, Son., Mexico.
Moulton, Herbert G.,	Cobalt, Ont., Can.
Muir, Thomas K.,	Portland, Ore.
Piper, John W. H.,	Buenos Ayres, Argentine Rep., S. A.
Potter, J. A.,	41 W. 124th St., New York, N. Y.
Sandifer, Harmer C.,	El Oro, Mexico.
Schlemm, William H.,	Durango, Mexico.
Scott, Winfield G.,	Long Beach, Cal.
Skelding, Joseph F.,	Embreeville, Tenn.
Thomas, Richard A.,	43 Wall St., New York, N. Y.
Vidler, Louis W.,	Lookout Mountain, Colo.
Warren, Henry L. J.,	Salt Lake City, Utah.
Wolfe, Burton L.,	Ely, Nev.
Young, William,	Kenora, Ont., Canada.

NECROLOGY.

The deaths of the following members have been reported to the Secretary's office during the month of November, 1909:

Date of Election.	Name.	Date of Decease.
1905.	*James W. Brown,	October 23, 1909.
1891.	*Emil Krabler,	October 24, 1909.

* Member.

BIOGRAPHICAL NOTICES.

James W. Brown was born July 14, 1844, at Pittsburg, Pa., and, after completing his preliminary education, entered in 1862 the iron business of his father, William R. Brown, one of the pioneers of that industry in Pittsburg. In 1866, he accepted a position with the Wayne Iron Works, and ten years later he became the assistant mill-manager of Hussey, Wells & Co., of which house he became, after successive promotions, a partner, the name of the firm being changed, first to Hussey, Brown & Co., and then to Howe, Brown & Co. When the Crucible Steel Co. of America absorbed this concern, Mr. Brown became a vice-president of the new organization. But in 1901 he retired, and formed the Colonial Steel Co., building a new plant at Colona, opposite Monoca, on the Monongahela river. Of this company he was President at the time of his death, which occurred suddenly, Oct. 23, 1909, at the house of the Pont Mouille Shooting Club, near Detroit, Mich., from acute indigestion inducing heart-failure. Mr. Brown had been a member of the Institute since 1905; and, though prevented by business duties from taking active part in its proceedings, heartily sympathized with its work and purpose. Apart from a single term in Congress, to which he was elected in 1902, and during which his knowledge of the iron and steel business was specially valuable in the shaping of legislation, he held no public office, though his fellow-citizens manifested in other ways, by employing him in positions of business responsibility, their esteem of him.

Emil Krabler was born Jan. 21, 1839, at Crossen on the Oder, in Germany. After preparatory education in sundry local schools, including the gymnasium at Aachen, he chose mining as his profession, and, having acquired by practice in the zinc- and coal-mines of Germany a preliminary training in that department, entered in 1859 the then new Mining Academy at Berlin. In 1861, he won in competition a prize of 250 thalers, and spent the money on a professional journey to Freiberg, Zwickau, etc., his report of which secured from the Prussian *Oberbergamt* his admission to the State service. From 1864 to 1871, he was occupied in various districts as a government official, promoted in rank from time to time; but in the

latter year he finally retired from that relation, to take charge, at Altenessen, near Cologne, of important private mining-enterprises, which involved the sinking of several deep shafts. Three years earlier, at the age of 29, he had assumed, under leave of absence, this responsibility; and his full administration of it was characterized by improvements in surface-plant and organization, as well as operations underground, which placed the undertaking on a profitable footing such as it had not enjoyed before. Moreover, by the building of model dwellings, hospitals, etc., for his employees, he sought to create a loyal body of workmen. But, like most others who have made this attempt,[1] he encountered the hostility of those "leaders" of "labor" who resent and oppose everything which tends to make the workman contented and indisposed to war with his employer; and so the great strike of 1872 stopped the enterprise in which Krabler was engaged, and forced him to seek elsewhere the employment which his ability and experience made it not difficult for him to find. After the successful fulfillment of several professional engagements, he was elected in 1886 a President of the Cologne *Bergwerksverein*, and in that capacity had to deal with the great miners' strike of 1889, the termination of which in favor of the mine-operators was largely due to his skillful and inspiring leadership.

In numerous other departments of the German mining industry (a more particular account of which may be found in *Stahl und Eisen*, Vol. xxix., No. 44, Nov. 3, 1909) Councilor Krabler achieved a high reputation as a technical expert and a business manager. The mining interests of the Cologne and Dortmund districts were specially indebted to him, and recognized their debt by the honors which they conferred upon him. Among numerous other such rewards may be mentioned the rank of Mining Councilor, given in 1903, and the extraordinary distinction of Privy Councilor, given in 1901, by the Prussian government.

In 1905, Councilor Krabler, as the head of the Dortmund Association for Mining Interests, had to deal with the great miners' strike of that year, and maintained successfully the principles with which his name was already identified. It is worthy of note that, although repeatedly involved in conflict with the so-called leaders of so-called labor, he was never in-

fluenced by the passions excited in such warfare to lose his interest in all movements which he regarded as promotive of the true interests of wage-earners, such as the establishment of benefit societies, savings banks, etc., to which he gave thought and labor without stint.

He was an influential member, and for many years an officer, of the Verein deutscher Eisenhüttenleute, and a member ot this Institute since 1901. He died Oct. 25, 1909, at Essen-Rüttenscheid, Germany.

SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

A New Separator for the Removal of Slate from Coal.

BY W. S. AYRES, HAZLETON, PA.

(Spokane Meeting, September, 1909.)

[SECRETARY'S NOTE.—At the Spokane meeting of the Institute, in discussion of President Brunton's address on "Modern Progress in Mining and Metallurgy in the Western United States," and at the request of members present, Mr. Ayres gave an oral account of his new separator, which is here published as an independent paper, partly because of its inherent importance and partly because it describes an improvement which did not originate in the Western United States, and therefore does not fall, strictly speaking, under the title of President Brunton's address.—R. W. R.]

A BRIEF history of the growth of the anthracite-coal preparation will give a better view-point from which to judge the present problem of separating slate from coal.

At the beginning of the commercial value of anthracite, 70 years ago, only the pure portions, or "splits," of the veins were mined and shipped to market, and without any preparation or screening other than the selection, while loading, of the glassy lumps, and the rejection of the fine material and the slate that had strayed accidentally into the coal. The next step was the crushing or breaking of the coal (hence the name "breaker" as applied to the preparation-building), and the sizing of it by means of bars or revolving screens. This stage of its development marked the advent of the "breaker-boy" as a slate-picker, with his ever-increasing capriciousness.

As the richer veins or "splits" became exhausted and the market demanded a still greater output, the less-pure "splits" and the thinner veins were utilized to produce the coal. Carrying as they do a far greater percentage of impurities, particularly when removing the pillars, it became necessary to build new and better equipped preparation-plants. Finally, we are now at the highest stage of complication yet known to the art of coal-preparation. We are dealing with varying specific gravities, frictional difference, hardness, structure, and form in the pieces of coal and slate coming from many different veins, and

all mixed together in different proportions. The treatment of each vein separately is, of course, impossible because of the size of the plant required. The re-treating of the refuse-banks from the early mining-operations is, however, generally done in an individual plant, termed a "washery," separate from that used for fresh-mined coal.

These complications in handling the material have brought forward several types of jigs and mechanical pickers, all of which are more or less wasteful of the coal.

A very careful study during the past 15 years of the conservation of coal after it has been delivered from the mine to the preparation-plant, or "breaker," has led me to the devising of means to prevent the very great losses sustained. These losses consist of undue chipping of the coal in the process of crushing, in the process of screening or sizing, in the jigs and other separating-machinery, and in the conveying-chutes, or "telegraphs," as they are called. Fundamentally, every impact destroys values by chipping from the larger pieces, which have the greatest value, very small particles which are practically valueless. These losses range from 1 to 2 per cent. in a right angle bend in a straight chute, or "telegraph," to 20 per cent. in a jig. In the preparation- or slate-picking machinery alone the losses range usually from 5 to 20 per cent. The picking of slate by hand is very wasteful also. The average boy throws out about as much coal as he does slate, and much more on dark days and at night.

In the construction of the coal-breaker at the Cranberry mine, at Hazleton, Pa., in 1896, I designed and introduced for the first time the continuous spiral chute, which is fully described in my paper, The New Breaker at Cranberry Coal-Mine.[1] This chute, having pitches determined by experiment for each size of coal, delivers the coal from the separating-machines to the pockets at a very moderate speed, and with a very decidedly smaller amount of loss in chippings than the ordinary straight chute. The saving is about 2.5 per cent. This chute is now quite extensively used throughout the region.

Material losses in the crushing-rolls and screens have been greatly reduced by improved types of these machines. In

[1] *Trans.*, xxviii., 293 (1898).

1896 I gave to a large manufacturing concern some data showing that the rolls should be at least 48 in. in diameter.

The greatest losses, however, are in the separating-machinery. All jigs are destructive because of the 80, more or less, impulses given the material per minute, thus continually grinding from the coal small particles which are valueless. All mechanical pickers heretofore designed are also very destructive, because the coal must sustain severe impacts in passing through them, and consequently create very great losses.

It is with a view to avoiding these losses that I have made many exhaustive tests at my testing-plant.

There are only two handles, so to speak, known to the art of separating slate from coal, by which we can mechanically take hold of the problem; one is difference in specific gravity, applied in jigging, and the other is frictional difference, or difference in the angle of repose, applied in frictional separators.

A somewhat better understanding of it may be had from a study of the material itself. Much of the material as it comes from the screens is composed of the following different forms of coal and slate, classified for convenience into six groups:

(1) Glassy fractured coal, usually cubical in form.

(2) Flat coal, some pieces having slate faces.

(3) Bone (interlaminated coal and slate), usually flat, and either coal-faced or slate-faced.

(4) Flat slate, from 0.25 to 0.5 in. thick.

(5) Pure slate with coal faces, approximately cubical in form.

(6) Slate and rock, heavy, and cubical in form.

The jig, in addition to the fault of seriously abrading the coal, does not effect a good separation when working on material composed of pieces having such widely-differing forms. In the coal-discharge is found a most unsatisfactory and aggravating mixture of slate and coal. Groups (1) and (2) predominate, but with them is found a large percentage of group (4)—flat slate—and a considerable quantity of groups (5) and (6)—pure slate and rock. In the slate-discharge, on the other hand, are found groups (5) and (6) predominating, with a high percentage of group (1)—glassy fractured cubical coal. The fact that not a single piece of coal taken from the slate-discharge has ever been found with a specific gravity greater than even the lightest piece of slate, is conclusive. Therefore the old and

oft-repeated explanation of this erratic action of jigs, viz., "that the specific gravity of some of the coal is greater than that of the slate," becomes a myth.

It is plain, therefore, that a jig is inefficient with this class of material, which now constitutes the greater part of that brought to the "breaker," and that the form-difference of the pieces has a dominating influence over the process of separation. In fact, the cause of this erratic action has been so well defined by classifying the pieces composing this class of material into form-groups and then determining individually their weights, areas exposed to the impulses of the jig, and their specific gravities, that the wonder is that the jig effects any separation at all. In fact, when certain groups predominate there is practically no separation. The cause lies in the fact that the area exposed to the impulses of the water does not have a uniform ratio to the cubic content of the pieces. Consequently, a piece of flat slate is sure to be lifted into the coal-zone, and, on the other hand, a cubical piece of coal can be dropped into the slate-zone. In fact, because of the shifting positions of these more or less flat or elongated pieces, from edgewise to flatwise to the impulses of the water, no distinct and progressive separation-zones are established in the jig; only a continuous mixing is the result. The ideal conditions for a jig are that the material shall be composed of spheres of the same diameter, and that the coal shall have a specific gravity that is lighter than that of the slate.

It is also well established that group (2)—flat coal with slate faces—and group (5)—pure slate with coal faces—cannot be separated by any frictional separating-machine; in fact, the jig is the only separating-device that will make the separation.

In view of the foregoing difficulties I have designed a separator that would, at least, meet the greatest of them. The chief objects in its design were (1) the avoidance of all impacts and the consequent wasteful chipping of the coal; (2) the doing away as far as possible with hand-picking; (3) the removing of flat slate without wasting the flat coal; (4) the exposing of the operation to view, so that the exact action of the machine could be observed at all times at a glance; and (5) the ability to adjust the machine while in operation.

Fig. 1 clearly shows the construction and operation of the machine.

The traveling separating-belt, mounted on two shafts, is made to move upwardly on its upper run by means of the drive-pulley. The belt, made of metal slats attached to a specially-designed link-belt, is inclined forwardly as well as transversely, and at such angles as are suitable for the proper separation of the material to be treated. Its transverse in-

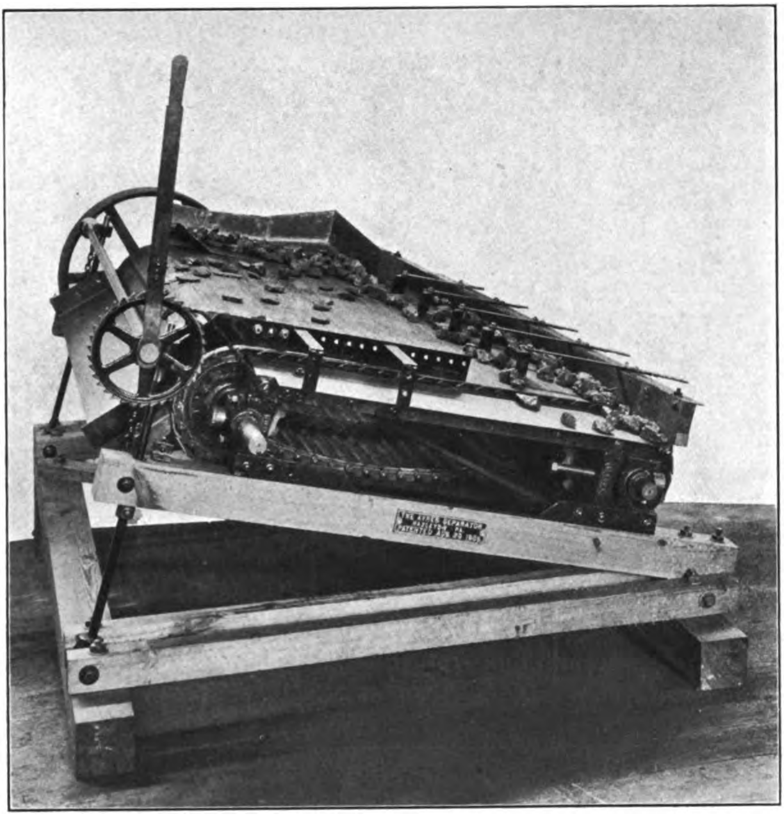

FIG. 1.—THE AYRES SEPARATOR.

clination is adjustable, while running, by means of the lever and ratchet-wheel.

The material is fed in a continuous stream upon the pan at the farther and higher end of the machine through a properly-constructed feed-chute. As it slides off of the pan the upwardly-moving belt immediately spreads the material out into a well-spaced stream, so that the coal may slide down against the guide and off of the lower right-hand corner of the machine,

while the slate, adhering to the belt, may move upwardly to the left-hand side and thus out of the forwardly-moving stream of coal.

The coal is conveyed away from the machine in any desired direction by a suitable chute, and the slate in like manner is collected along the slate-apron at the left in a similar chute and disposed of as desired.

The capacity of the machine, which varies somewhat with the nature of the material, but chiefly with the size of the coal, ranges from 5 tons per hour on pea size (through a 0.75- and over a 0.5-in. mesh), to 25 tons per hour on steamboat size (through a 6- and over a 4.5-in. mesh). The operation is distinctively open to view.

The saving by preventing the loss in chipping, which, as already stated, ranges in other separators from 5 to 20 per cent. of the coal treated, is enormous. The chippings on this new separator amount, on an average, to less than 0.5 per cent. On the low basis of 5 per cent. saved, the amount would be $18,750 on every 100,000 tons of prepared sizes shipped. One installation of 10 machines, now in operation for more than a year, shows a gain of 19.5 per cent. in the prepared sizes, with an output of 11,500 tons per month, or a gain of $4,152 per month. In addition to the gain by the prevention of chippings, a saving in labor of $500 per month has been effected.

The many installations on steamboat size (which is usually hand-picked) show a very great saving in labor, each machine doing the work of from 5 to 12 men, besides giving a decidedly more uniformly clean product, which, when crushed to prepared sizes, as is now almost universally done, reduces the labor still further on the preparation after crushing.

It is not possible in all cases to do away entirely with hand-picking by the use of the separator, because of the occasional presence of pieces of slate having coal faces, and pieces of coal having slate faces, but instead of having, as in one case, 32 men hand-picking on steamboat size, now 4 of these separators and 12 men do the work. The period of time required to pay for the equipment by its actual saving ranges from 10 to 80 days.

In treating such material as has been described in connection with the jig, it has been found most effective to use a three-

stage process. The first process to be a set of these separators, turning direct to the pocket, without appreciable loss from chippings, 60 to 75 per cent. of the total coal in the material, which includes all the glassy fractured cubical coal—group (1)—and all of the flat coal not having slate faces. The second process to be likewise a set of these separators so adjusted as to remove all the flat slate—group (4)—and all pure slate and rock—group (6). With the foregoing mixture of forms trimmed, so to speak, in this way, we have left group (2), flat coal with slate faces; group (3), bone; and group (5), pure slate with coal faces. Passing this product, which contains only from 25 to 40 per cent. of the total coal, to a jig as the third process, a very satisfactory separation can be made.

By this three-stage process the losses have been reduced to a minimum, or, to put it the other way, the gains as shown in the installation of the 10 machines previously mentioned have been as high as 19.5 per cent. in the prepared sizes.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Influence of Top-Lag on the Depth of the Pipe in Steel Ingots.

BY HENRY M. HOWE,* NEW YORK, N. Y.

(Spokane Meeting, September, 1909.)

IN my original paper, Piping and Segregation in Steel Ingots, I pointed out[1] among other things that, in view of the slighter stretching (virtual expansion) of the crust, and greater opportunity for sagging, there should be less piping in broad than in narrow ingots, and less in slowly-cooled ingots, *e.g.*, those cast in pre-heated sand molds, than in those which cool quickly, *e.g.*, those cast in iron molds. A. A. Stevenson[2] said that neither of these predictions agreed with his own experience. In particular, in a picture which he showed of a wide and of a narrow ingot cast from the same ladleful of steel,[3] the wide ingot had certainly piped much more deeply than the narrow one.

At the time I did not see the explanation of these discrepancies, but further reflection makes it evident.

One of the most important elements in determining the depth of the pipe is the degree of "top-lag," that is, the degree to which the solidification of the top of the ingot lags behind that of the bottom. Through this lagging the steel of the upper part of the ingot is able to run down and fill the pipe below as fast as it forms. To the importance of this lagging I called attention in my original paper.[4]

Sand vs. Iron Molds.—If we compare two like ingots, one cast in an iron and the other in a sand mold, we see that the top-lag is much greater in the former than in the latter, because in the former the lower part of the ingot is cooling off fast while the metal is running into the upper part. It is perhaps easier to look at this as a case of the solidification of the bot-

* Professor of Metallurgy, Columbia University, New York, N. Y.

[1] *Trans.*, xxxviii., 56 to 58 (1908).
[2] *Idem*, xxxix., 834 to 836 (1909).
[3] *Idem*, 837.
[4] *Idem*, xxxviii., 58.

tom outrunning that of the top, which is nothing but the other aspect of top-lag. This stronger top-lag in case of iron than in case of sand molds may well outweigh the influence of greater opportunity for sagging which the sand mold gives. It is to this effect that I refer the discrepancy between Mr. Stevenson's observation and my prediction. The latter ought to have been modified so as to take into account the greater top-lag in the iron mold. If this influence can be cut out, then the effect of greater opportunity for sagging in the sand mold should become evident in the shortening of the pipe. In experiments which I have since tried I have found this to be the case.

A striking example of the shortening effect of slow cooling, which, as I asserted, ought to shorten the pipe, is given in the case of ingots which solidify slowly in the soaking-pit. Their pipe is much shorter than that of ingots which solidify rapidly in the outer air.

Wide vs. Narrow Ingots.—The case which Mr. Stevenson gives, in which a narrow ingot piped much less deeply than a wide one cast from the same ladleful of steel, is seen, on further consideration, not to be a fair contradiction of my prediction, for two reasons. In the first place, the fact that these two ingots were cast with the wide end up tends to shorten the pipe much more in the narrow than in the wide ingot. I have insisted on the effect which having the large end up has of shortening the pipe by means of top-lag,[5] though I had not at that time devised this term. It is clear that the effect of this taper is much greater in a narrow than in a wide ingot. The taper is usually the same, and hence the absolute widening of the top is the same, in narrow as in wide ingots, and hence it forms a much larger proportion of the width of the ingot in narrow than in wide ingots. But the mere fact that the widening at the top bears a greater proportion to the average width of the ingot in narrow than in wide ingots has for its clear result that this widening causes more top-lag in narrow than in wide ingots. The effect of width as such on the depth of the pipe can be shown only when the effect of other variables is cut out. Now in this case the greater top-lag caused by the taper in the narrow than in the wide ingot directly opposed the effect of width as such in permitting sagging and in lessening crust-stretch. In order to test

[5] *Idem*, 60.

the effect of width as such, parallel-sided ingots should be used, and the effect of other variables should be excluded. This I have done in certain preliminary experiments, which, as far as they go, support my prediction that width tends to shorten the pipe.

In case the ingots are tapered in the opposite direction, with the large end down, this taper, because it tends to lengthen the pipe, and because the effect of taper should be inversely proportional to the width of the ingot, should tend to lengthen the pipe more in narrow than in wide ingots. In fact, this influence is relatively unimportant in wide ingots.

The second reason why the evidence given by Mr. Stevenson's wide and narrow ingots is not valid is that the narrow ingot was poured much more slowly than the wide one, and this in itself, as I pointed out clearly,[6] has an important effect in shortening the pipe. It was evidently poured much more slowly than the wide ingot, because the two were in the same bottom-cast group, and consequently the steel must have entered the narrow ingot very much more slowly than the wide one.

Everything else being equal, the more-rapid cooling of the bottom of a narrow than of a wide ingot tends to give the former greater top-lag than the latter, and thus to shorten its pipe.

Looking at it in a general way, we see that narrowness in one way tends to shorten the pipe and in other ways tends to lengthen it. On one hand, in that it leads to (1) the more-rapid cooling of the bottom, it increases top-lag and thereby tends to shorten the pipe. On the other hand, narrowness tends to lengthen the pipe (2) by leading to relatively great crust-stretching (virtual expansion), (3) by giving little opportunity for sagging, and (4) (for given rate of pouring) by leading to rapid rise of the surface of the metal, and in this way lessening the top-lag. My original prediction, supported by the observations which I had then made, was based on these latter considerations, (2), (3), and (4), and overlooked consideration (1). Now it may be shown hereafter that my prediction does not hold true under certain comparable conditions, or even under most comparable conditions. But Mr. Stevenson's

[6] *Idem*, 60.

evidence does not prove this, because if my prediction is true that the net effect of narrowness as such is to lengthen the pipe, nevertheless this effect might be completely masked under his conditions by the joint effect of (1) his having the large end up and (2) his pouring more rapidly into the wide than into the narrow ingot, because both of these things should tend to give the narrow ingot the shorter pipe of the two.

In other words, his conditions introduced certain accidental concomitants of narrowness, which concomitants clearly tend to shorten the pipe in his narrow ingots, and thus to mask the influence of narrowness as such. The fact that in the presence of these pipe-shortening concomitants the predicted pipe-lengthening effect of narrowness as such is not seen, is no proof either that that effect does not exist or that it would not be seen when such masking concomitants are absent.

I hope to present further data on this subject soon.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Proceedings of the Ninety-Seventh Meeting, Spokane, Wash., September, 1909.

COMMITTEES.

BUTTE, MONT.—Charles W. Goodale, *Chairman;* B. H. Dunshee, Benjamin B. Thayer, John Gillie, William Scallon, C. F. Kelly, H. A. Gallway, A. C. Carson, A. H. Wethey, John C. Adams, Oscar Rohn.

ANACONDA, MONT.—E. P. Mathewson, *Chairman;* William Wraith, J. C. Guinness.

CŒUR D'ALENE DISTRICT, IDAHO.—Roy H. Clarke, *Chairman;* Frederick Burbidge, Stanley Easton, J. F. McCarthy, H. L. Day, J. V. Richards, D. L. Huntington, J. B. Fisken.

SPOKANE, WASH.—E. J. Roberts, *Chairman;* L. K. Armstrong, *Secretary;* Charles P. Robbins, C. M. Fassett, J. C. Ralston, J. C. Haas, J. V. Richards, Roy H. Clarke, W. C. Miller, R. Marsh, J. M. Porter. Also, Western Branch of the Canadian Mining Institute, Thomas Kiddie, *Chairman;* E. Jacobs, *Secretary;* W. Fleet Robertson.

SEATTLE, WASH.—Frank A. Hill, *Chairman;* Chester F. Lee, *Secretary;* Milnor Roberts, M. K. Rodgers, F. A. Thomson.

TACOMA, WASH.—John H. Williams, *Chairman;* C. R. Claghorn, Henry Hewitt, Jr., Charles A. Foster, F. W. Clark, Roger Taylor, John N. Pott.

SALT LAKE CITY, UTAH.—Duncan MacVichie, *Chairman;* R. H. Bradford, J. M. Callow, Ellsworth Daggett, E. E. Nelson, R. S. Oliver, R. Forrester, C. C. Crimson, H. R. Ellis, Samuel Newhouse.

BINGHAM, UTAH.—D. C. Jackling, L. Hanchett, J. C. Dick, C. H. Doolittle, R. C. Gemmell, L. S. Cates, E. P. Jennings, M. M. Johnson, Samuel Newhouse, R. Forrester, G. Lavagnino, C. W. Saxman, B. F. Tibby, E. A. Wall, Thomas Weir, J. B. Risque, C. E. Allen, G. W. Metcalf, S. R. Woodbridge.

GARFIELD, UTAH.—A. J. Bettles, L. Hanchett, F. G. Janney, D. C. Jackling, Samuel Newhouse, R. Forrester, C. W. Whitley, P. B. Tracy, R. C. Gemmell.

BINGHAM JUNCTION AND MURRAY, UTAH.—C. W. Whitley, George W. Heintz, P. E. Barbour.

PARK CITY, UTAH.—George D. Blood, E. W. Durfee, F. W. Sherman, F. T. Williams.

TINTIC DISTRICT, UTAH.—P. J. Donohue, J. H. McCrystal, I. N. Dunyon, J. C. McCrystal, G. W. Riter, Col. C. E. Loose, C. E. Allen.

PUEBLO, COLO.—J. B. McKennan, F. E. Parks, H. A. Deuel, F. Guiterman, George A. Marsh, B. Hogarty, C. R. Rose, A. Stock.

The Institute Headquarters at Spokane was established at the Spokane Hotel, and included a Bureau of Information for the benefit and comfort of members and guests of the party during the time of the meeting.

The first session, held Tuesday afternoon, Sept. 28, 1909, in the beautiful Convention Hall of Masonic Temple, was called to order by Mr. E. J. Roberts, Chairman of the Spokane Local Committee, who introduced Mr. J. C. Ralston, City Engineer, representing the Mayor of Spokane and the Spokane Chamber of Commerce. Mr. Ralston extended to the members and guests of the Institute a cordial welcome to the city of Spokane, substantially as follows:

Mr. President and Members of the American Institute of Mining Engineers:

Ferdinand and Isabella were the first to initiate mining on the Western Hemisphere when they grub-staked Columbus. Since that time mining has gone forward and grown into the best and most approved methods and practice, gradually coming under the influence of the efficient methods and talent of the American Institute of Mining Engineers.

If the American Institute of Mining Engineers stands for anything, it stands for progressiveness and efficiency; but pre-eminently for "doing things." Collectively and individually you are a body of men who, from the inception of the Institute, have been the forerunners of every effort which has brought mining on the American continent up to its high technical and economic standing. By reason of the fact that you are so conspicuously "doing things," it gives Spokane pleasure, through me as its representative, to welcome you to our city.

Spokane believes itself to be a municipality and a community which also is "doing things," and a kindred and common feeling, therefore, must of necessity exist. It is therefore with all the more propriety that Spokane bids you welcome.

Last year some gentleman in this city suggested to the Chamber of Commerce that it institute an apple show, at which the apple product of the Spokane valley should be exhibited. Within a week thereafter another gentleman with more boldness suggested that it be made a State affair. This met with instant approval. Within a few days again still another gentleman with more temerity than the others suggested that it be made national. The result was that it was made a National Apple Show. The conspicuous success of this enterprise within the first year has amply justified the judgment of all concerned. This is merely an example of how, in one of many details, Spokane is "doing things."

Within the past two months there has been held in Spokane a National Irrigation Congress, at which there were more than 1,500 accredited delegates from all parts of the United States. The gross estimated annual value of all products due to irrigation within the territory tributary to Spokane—namely, the Inland Empire—amounted to about $14,000,000. We therefore had 1,500 delegates representing, so far as Spokane is concerned, a gross product of $14,000,000.

I am glad to have the opportunity at this time to make a comparison between the products of irrigation and the products of the mining industry, in what may be called Spokane territory. I see seated before me less than 150 delegates to this Institute meeting. It may startle you when I inform you that the estimated gross annual value of the mining-products derived from the territory tributary to Spokane amounts to about $40,000,000, which is nearly three times as great an amount as that derived from irrigation, though the number of your delegates is only one-tenth as large in this case as that of the representatives of the ancient profession of irrigation. The parallel is striking and furnishes a marked commentary upon

the fact that the American Institute of Mining Engineers stands for very substantial interests, and is "doing things."

The Chamber of Commerce of this city is perhaps one of the most efficient and aggressive commercial bodies in the United States. The work which it has done in the past few years speaks for itself in the most impressive manner. It is a force and an asset of a very substantial nature of the city of Spokane.

It is therefore peculiarly fitting that the present meeting of the Institute should be held in this city, and it gives me great pleasure, by reason of the personalities and potentialities which I here represent, to extend to you, on behalf of our distinguished Mayor and the Chamber of Commerce, a most cordial welcome.

Mr. Ralston's address was acknowledged by President David W. Brunton, and responded to by the Secretary, Dr. R. W. Raymond.

The following paper was presented in oral abstract by the author:

*Modern Progress in Mining and Metallurgy in the Western United States, by David W. Brunton, Denver, Colo.

This paper was discussed by W. S. Ayres, Hazleton, Pa. (coal-recovery); Prof. William Kent, Sandusky, O. (debt of mining engineering to the mechanical engineer); Charles Catlett, Staunton, Va. (cost-keeping and efficiency-records); Dr. W. O. Snelling, Pittsburg, Pa. (explosions); E. S. Hutchinson, Newtown, Pa. (general); Charles W. Goodale, Butte, Mont. (dust-recovery); Ernest Levy, Rossland, B. C. (replacement-mining); W. L. Saunders, New York, N. Y. (air-compression).

The second session of the Institute, held at the same place, Tuesday evening, Sept. 28, was called to order by President Brunton.

The following papers were presented in oral abstract by the authors:

Discussion (continued) of Mr. Brunton's paper, Modern Progress in Mining and Metallurgy in the Western United States, by Thomas Kiddie, Northport, Wash. (fume-condensation).

*Modern Practice of Ore-Sampling, by David W. Brunton, Denver, Colo.

In the absence of the author, the Secretary presented in oral abstract the following paper:

*Dust-Explosions in Coal-Mines, by Franklin Bache, Fort Smith, Ark. Discussed by Dr. R. W. Raymond.

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* Printed copies weie available for distribntion.

The following paper was presented in oral abstract by the author:

Causes of Variation in Ore-Sampling, by William Kiddie, Northport, Wash.*

Under a mutual agreement between the Councils of the American Institute of Mining Engineers and the Western Branch of the Canadian Mining Institute, it was decided to hold a joint session of both Institutes during the meeting at Spokane. This agreement also provided that each of the two societies should be free to publish such papers and discussions, presented at this session, as it should deem desirable, and accordingly the text of the papers presented by the Western Branch of the Canadian Mining Institute will be found either in forthcoming numbers of the *Bulletin*, or in the *Proceedings of the Canadian Mining Institute.*

The third and concluding session of the Institute was selected as best suited for the joint meeting.

This session, held at the same place, Wednesday morning, Sept. 29, was presided over by President Brunton, of the American Institute of Mining Engineers, and President Kiddie, of the Western Branch of the Canadian Mining Institute.

The following papers were presented in oral abstract by the authors:

†The Ruble Hydraulic Elevator, by J. McD. Porter, Spokane, Wash.

The Conservation of Coal in the United States, by E. W. Parker, Washington, D. C. Discussed by Dr. R. W. Raymond and Prof. William Kent.

Coal-Mining in Southeastern British Columbia and in Alberta, by E. Jacobs, Victoria, B. C.

In the absence of the author, the following paper was presented in oral abstract by Mr. Jacobs:

The Galt Coal-Field, Lethbridge, Alberta, by W. D. L. Hardie. Discussed by W. S. Ayres, Hazleton, Pa. (Galt and Bankhead coal-fields); Milnor Roberts, Seattle, Wash. (Nicola fuel-region); Wm. Fleet Robertson, Victoria, B. C. (North Vancouver Island and Queen Charlotte Island fields); Charles

* Published in abstract in *Engineering and Mining Journal*, vol. 88, No. 17, pp. 825–826 (Oct. 23, 1909).

† Printed copies were available for distribution.

Catlett, Staunton, Va. (coal-prices); Frederick Keffer, Greenwood, B. C. (coal- and coke-prices at furnaces); and William Kiddie, Northport, Wash. (general).

In addition to the papers already noted, the following were read by title for future publication:

*The Formation and Enrichment of Ore-Bearing Veins, by George J. Bancroft, Denver, Colo.

*Need of Instrumental Surveying in Practical Geology, by Benjamin Smith Lyman, Philadelphia, Pa.

*The Assay and Valuation of Gold Bullion, by Frederic P. Dewey, Washington, D. C.

The Cyaniding of Silver-Ores in Mexico, by Albert F. J. Bordeaux, Thonon les Bains, France.

The Fushun Colliery, South Manchuria, by Warden A. Moller, Tientsin, China.

Protective Value of Humidity in Dusty or Gaseous Coal-Mines, by James Ashworth, Congleton, England.

*The Ventilating System of the Comstock Mines, Nevada, by George J. Young, Reno, Nev.

Professional Ethics, by Victor G. Hills, Denver, Colo.

An Adjustable Pyrometer-Stand, by L. W. Bahney, Palo Alto, Cal.

Glass Mine-Models, by Edmund D. North, Tonopah, Nev.

*Postscript to paper, The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes, by H. O. Hofman and W. Mostowitsch, Boston, Mass.

Cyaniding Slime, by Mark R. Lamb, Milwaukee, Wis.

*Influence of Ingot-Size on the Degree of Segregation in Steel Ingots, by Henry M. Howe, New York, N. Y.

Preparing and Recording Samples for Use in Technical Assay-Laboratories, by Louis D. Huntoon, New Haven, Conn.

Dredging for Gold in French Guiana, by Albert F. J. Bordeaux, Thonon les Bains, France.

Mining Industry of Nicaragua, by T. Lane Carter, Bluefields, Nicaragua, Central America.

Barite Industry of the United States, by A. A. Steel, Fayetteville, Ark.

Federal Coal-Mines in the Philippines, by Oscar H. Reinholt, Pasadena, Cal.

* Printed copies were available for distribution.

Discussion of the paper of Charles R. Keyes, Genesis of the Lake Valley, New Mexico, Silver-Deposits, by William M. Courtis, Detroit, Mich.

Discussion of the paper of Henry M. Howe, Piping and Segregation in Steel Ingots, by P. H. Dudley, New York, N. Y.

Discussion of the paper of Charles R. Keyes, Ozark Lead- and Zinc-Deposits; Their Genesis, Localization, and Migration, by E. R. Buckley, Flat River, Mo.

Discussion of the paper of D. F. Hewett, Vanadium-Deposits in Peru, by James F. Kemp, New York, N. Y.

Discussion of the paper of Hofman and Hayward, Pan-Amalgamation; an Instructive Laboratory-Experiment, by E. A. H. Tays, San Blas, Sinaloa, Mexico.

*The Influence of Bismuth on Wire-Bar Copper, by H. N. Lawrie, Portland, Ore.

The Barometric and Temperature Conditions at the Time of Dust-Explosions in the Appalachian Coal-Mines by N. H. Mannakee, Williamson, W. Va.

Influence of Top Lag on the Depth of the Pipe in Steel Ingots, by H. M. Howe, New York, N. Y.

The Combustion-Temperature of Carbon and Its Relation to Blast-Furnace Phenomena, by Clarence P. Linville, State College, Pa.

*The Limit of Fuel-Economy in the Iron Blast-Furnace, by N. M. Langdon, Mancelona, Mich.

*Borax-Deposits in the United States, by Charles R. Keyes, Des Moines, Iowa.

*Conditions and Costs of Mining at the Braden Copper-Mines, Chile, by William Braden, New York, N. Y.

* Printed copies were available for distribution.

Excursions and Entertainments.

At 9 a.m., on Thursday, Sept. 16, the special train conveying the Institute party left the Union Station at Chicago by the Chicago, Milwaukee & St. Paul railway. The train consisted of three Pullman sleepers (the *Blythedale*, the *Salvador*, and the *Wandermere*), the Pullman dining-car *Victoria*, and a baggage-car. The party comprised at starting about 60 persons, but was subsequently augmented from time to time *en route*.

The following list contains the names of those constituting the special excursion party:

Ayres, W. S., Hazleton, Pa.
Ayres, Mrs. W. S., Hazleton, Pa.
Bellinger, A. R., Syracuse, N. Y.
Bostwick, F. H., Denver, Colo.
Bostwick, Mrs. F. H., Denver, Colo.
Brooke, D. Owen, Birdsboro, Pa.
Brooke, Mrs. D. Owen, Birdsboro, Pa.
Brunton, D. W., Denver, Colo.
Catlett, Charles, Staunton, Va.
Chamberlain, H. S., Chattanooga, Tenn.
Chamberlain, Mrs., Chattanooga, Tenn.
Dougherty, J. W., Steelton, Pa.
Douglas, Miss J. L., Brooklyn, N. Y.
Fries, Miss Anna, Philadelphia, Pa.
Glendenning, Miss J. A., New York, N.Y.
Greenfield, T. B., London, Eng.
Harrington, Arthur, Philadelphia, Pa.
Harrington, Miss Helen, Phila., Pa.
Harrington, M. H., Philadelphia, Pa.
Harrington, Mrs. M. H., Phila., Pa.
Hutchinson, E. S., Newtown, Pa.
Hutchinson, Mrs. E. S., Newtown, Pa.
Jones, T. D., Hazleton, Pa.
Kanda, Reiji, Tokyo, Japan.
Kelly, Wm., Vulcan, Mich.
Kelly, Mrs. Wm., Vulcan, Mich.
Kent, Wm., Sandusky, Ohio.
Kent, Mrs. Wm., Sandusky, Ohio.
Lawton, A. H., New York, N. Y.
Lilly, John, Lambertville, N. J.
Lilly, Mrs. John, Lambertville, N. J.
Lilly, Wm., Lambertville, N. J.
MacFarland, Mrs. F. L., Denver, Colo.
McCrery, Charles, Birmingham, Ala.
Mitchell, W. S, Haileybury, Ont., Can.
Mitchell, Mrs. W. S., Haileybury, Can.
Nesmith, J. W., Denver, Colo.
Nesmith, Mrs. J. W., Denver, Colo.
Parker, E. W., Washington, D. C.
Perry, J. G., Denver, Colo.
Perry, Mrs. J. G., Denver, Colo.
Pilling, Ross, Philadelphia, Pa.
Pilling, W. S., Philadelphia, Pa.
Pilling, Mrs. W. S., Philadelphia, Pa.
Pilling, Miss M. B., Philadelphia, Pa.
Pinkney, H. H., Macdonald, W. Va.
Pitkin, S. H., Cleveland, Ohio.
Raymond, Dr. R. W., New York, N. Y.
Saunders, Miss Emily, Philadelphia, Pa.
Saunders, Miss Jean, New York, N. Y.
Saunders, Miss Louise, New York, N. Y.
Saunders, Mrs. W. B., Philadelphia, Pa.
Saunders, W. L., New York, N. Y.
Saunders, W. L., Jr., Philadelphia, Pa.
Shurick, A. T., Washoe, Mont.
Smith, Mrs. T. B., Birdsboro, Pa.
Snelling, Dr. W. O., Pittsburg, Pa.
Steiger, Geo., Washington, D. C.
Struthers, Dr. Joseph, New York, N. Y.
Vaughan, A. E., New York, N. Y.
Vaughan, Mrs. A. E., New York, N. Y.
Wellman, S. T., Cleveland, Ohio.
Weiss, C. R., Philadelphia, Pa.

No regular registry was made at Spokane or elsewhere; but numerous members joined the party from time to time, as the following incomplete list will testify:

Adair, Jas. B., Seattle, Wash.
Adair, Miss Minor G., Seattle, Wash.
Adams, Mrs. J. C., Butte, Mont.
Adams, M. A., Seattle, Wash.
Adams, Wm. H., Seattle, Wash.
Adams, Wm. H., Jr., Seattle, Wash.
Adams, John C, Butte, Mont.
Allen, C. E., Bingham, Utah.
Anderson, James, Seattle, Wash.
Armstead, Mrs. D. M., Butte, Mont.
Armstrong, L. K., Spokane, Wash.
Austin, Leonard S., Salt Lake, Utah.
Austin, W. Lawrence, Riverside, Cal.
Auzias, Turenne R., Seattle, Wash.
Barbour, P. E., Murray, Utah.
Bennette, Nelson, Tacoma, Wash.
Bettles, A. J., Garfield, Utah.
Blair, A. F., Seattle, Wash.
Blood, George D., Park City, Utah.
Bogardus, C. E., Seattle, Wash.
Boykin, James C., Seattle, Wash.
Bradford, R. H., Salt Lake City, Utah.
Burbidge, Fred'k, Silver King, Idaho.
Caetani, Golascio, Wardner, Idaho.
Caldwell, F. M., Seattle, Wash.
Callow, J. M., Salt Lake City, Utah.
Cambier, A., Pueblo, Colo.
Carroll, Eugene, Butte, Mont.
Carroll, Mrs. Eugene, Butte, Mont.
Carson, A. C., Butte, Mont.
Cates, L. S., Bingham, Utah.
Champion, John R., Leadville, Colo.
Claghorn, C. R., Tacoma, Wash.
Claghorn, Mrs. C. R., Tacoma, Wash.
Clarke, Roy H., Spokane, Wash.
Clark, F. W., Tacoma, Wash.
Cartwright, Miss A., Seattle, Wash.
Cozzens, Harmon, Pueblo, Colo.
Crimson, C. C., Salt Lake City, Utah.
Curtis, Miss Frank W., Seattle, Wash.
Daggett, Ellsworth, Salt Lake City, Utah.
Day, H. L., Spokane, Wash.
Deuel, H. A., Pueblo, Colo.
Dick, J. C., Bingham, Utah.
Donohue, P. J., Salt Lake, Utah.
Doolittle, C. H., Bingham, Utah.
Dunshee, B. H., Butte, Mont.
Dunshee, Mrs. B. H., Butte, Mont.
Dunyon, I. N., Salt Lake, Utah.
Durfee, E. W., Park City, Utah.
Easley, George R., Baker City, Ore.
Easton, Stanley, Wardner, Idaho.
Ellis, H. R., Salt Lake City, Utah.
Elmendorf, Mr.
Emmens, N. W., Trout Lake, B.C., Can.
Emmons, C. D., Eugene, Ore.
Evans, George W., Seattle, Wash.
Falkenburg, M. T., Seattle, Wash.
Farmer, Jas. L., Seattle, Wash.
Fassett, C. M., Spokane, Wash.
Fisken, J. B., Post Falls, Wash.
Forrester, R., Salt Lake City, Utah.
Foster, Charles A., Tacoma, Wash.
Gallway, H. A., Butte, Mont.
Gemmell, R. C., Bingham, Utah.
Gillie, John, Butte, Mont.
Gillie, Mrs. John, Butte, Mont.
Goodale, Charles W., Butte, Mont.
Goodsell, C. H., Spokane, Wash.
Griswold, C. T., Colorado Springs, Colo.
Guinness, J. C, Anaconda, Mont.
Guiterman, F., Pueblo, Colo.
Haas, J. C., Spokane, Wash.
Haas, Mrs. J. C., Spokane, Wash.
Hanchett, L., Bingham, Utah.
Heintz, George W., Murray, Utah.
Hewitt, Henry, Jr., Tacoma, Wash.
Hill, Frank A., Seattle, Wash.
Hill, Mrs. Frank A., Seattle, Wash.
Hilson, Cleaveland, Electric, Mont.
Hough, W. B., Wardner, Idaho.
Hodges, A. B. W., Grand Forks, Can.
Hodges, Mrs. A. B. W., Grand Forks, Can.
Hogarty, B., Pueblo, Colo.
Huntington, D. L., Post Falls, Wash.
Jackling, D. C., Bingham, Utah.
Jacobs, E., Victoria, B. C., Canada.
Jamme, George, Seattle, Wash.
Jamme, Mrs. George, Seattle, Wash.
Janney, F. G., Garfield, Utah.
Jennings, E. F., Bingham, Utah.
Johnson, M. M., Bingham, Utah.
Keffer, Frederic, Greenwood, B.C., Can.
Keffer, Mrs. Fred., Greenwood, B.C., Can.

Kelly, C. F., Butte, Mont.
Kelly, Mrs. C. F., Butte, Mont.
Kennedy, N. H., Spokane, Wash.
Kiddie, Thomas, Northport, Wash.
Knight, E. C., Seattle, Wash.
Knight, Mrs. E. C., Seattle, Wash.
Labarthe, J., Trail, B. C., Canada.
Labarthe, Mrs. J., Trail, B. C., Canada.
Landes, Mrs. Henry, Seattle, Wash.
Lavignino, G., Bingham, Utah.
Lee, Chester F., Seattle, Wash.
Lee, Mrs. Chester F., Seattle, Wash.
Levy, Ernest, Rossland, B. C., Canada.
Lewis, Clancy M., Seattle, Wash.
Lewis, Mrs. Clancy M., Seattle, Wash.
Loose, Col. C. E., Salt Lake, Utah.
Luckenbel, J. C., Spokane, Wash.
McCarthy, J. F., Burke, Idaho.
McCaustland, E J., Seattle, Wash.
McCaustland, Mrs. E. J., Seattle, Wash.
McClelland, J., Salt Lake, Utah.
McCrea, W. S., Spokane, Wash.
McCrystal, J. C., Salt Lake, Utah.
McKennan, J. B., Pueblo, Colo.
MacVichie, Duncan, Salt Lake, Utah.
Magnussen, C. E., Seattle, Wash.
Mannheim, P. A. L., New York, N. Y.
Marsh, R., Spokane, Wash.
Marsh, George A., Pueblo, Colo.
Mathewson, E. P., Anaconda, Mont.
Merrill, Mr., Salt Lake, Utah.
Metcalf, G. W., Bingham, Utah.
Miller, W. C., Spokane, Wash.
Moss, Milton, Huntsville, Ala.
Munson, H. C., Spokane, Wash.
Moore, Mrs. C. H., Butte, Mont.
Morony, Mrs. J. G., Butte, Mont.
Nelson, E. E., Salt Lake City, Utah.
Nevin, Charles, Butte, Mont.
Newhouse, Samuel, Salt Lake City, Utah.
Oliver, R. S., Salt Lake City, Utah.
Parks, F. E., Pueblo, Colo.
Phœnix, C. E., Tacoma, Wash.
Phœnix, Mrs. C. E., Tacoma, Wash.
Porter, J. McD., Spokane, Wash.
Pott, John N., Tacoma, Wash.
Pott, Mrs. John N., Tacoma, Wash.
Raht, August, Denver, Colo.
Rankin, W. J., Jr., Seattle, Wash.
Ralston, J. C., Spokane, Wash.
Ralston, Mrs. J. C., Spokane, Wash.
Richards, J. V., Spokane, Wash.
Richardson, S. H., Republic, Wash.
Risque, J. B., Bingham, Utah.
Riter, G. W., Salt Lake, Utah.
Robbins, Charles P., Spokane, Wash.
Roberts, E. J., Spokane, Wash.
Roberts, Milner, Seattle, Wash.
Roberts, Miss Milnora, Seattle, Wash.
Robertson, Wm. Fleet, Spokane, Wash.
Robertson, Mrs. W. Fleet, Victoria, Can.
Robertson, Douglas, Victoria, B.C., Can.
Rogers, M. K., Seattle, Wash.
Rogers, Master, Seattle, Wash.
Rohn, Oscar, Butte, Mont.
Rohn, Mrs. Oscar, Butte, Mont.
Rose, C. R., Pueblo, Colo.
Sanders, Wilbur E., Los Angeles, Cal.
Saxman, C. W., Bingham, Utah.
Scallon, William, Butte, Mont.
Sherman, F. W., Park City, Utah.
Smith, Jos. F., Salt Lake, Utah.
Stock, A., Pueblo, Colo.
Taylor, H. S., Chicago, Ill.
Taylor, Roger, Tacoma, Wash.
Thayer, Benjamin B., Butte, Mont.
Thomson, F. A., Seattle, Wash.
Tibby, B. F., Bingham, Utah.
Todd, Miss J., Seattle, Wash.
Tolman, L. P., Seattle, Wash.
Tracy, P. B., Garfield, Utah.
Trewarthe, James W. H., Victoria, Can.
Van Densen, Mrs., Seattle, Wash.
Weston, Samuel P., Seattle, Wash.
Wall, E. A., Bingham, Utah.
Weir, Thomas, Bingham, Utah.
Wethey, A. H., Butte, Mont.
Wharton, J. R., Butte, Mont.
White, Richard M., Seattle, Wash.
Whitley, C. W., Garfield, Utah.
Williams, F. T., Park City, Utah.
Williams, John H., Tacoma, Wash.
Williams, Mrs. John H., Tacoma, Wash.
Williams, Percy, Vancouver, B. C.
Woodbridge, S. R., Bingham, Utah.
Wraith, William, Anaconda, Mont.

The accompanying itinerary gives the places and routes concerned on the journey.

Itinerary.

Thursday,	Sept.	16th......	Lv.	Chicago...........9.00	A. M.	via C. M. & St. P.
Thursday,	"	16th......	Arr.	St. Paul...........9.40	P. M.	via C. M. & St. P.
Thursday,	"	16th......	Lv.	St. Paul about 10.00	P. M.	via N. P. R. R.
Friday,	"	17th......	En Route.............			via N. P. R. R.
Saturday,	"	18th......	Arr.	Yellowstone Park		via N. P. R. R.
Sunday,	"	19th......	In	Yellowstone Park		
Monday,	"	20th......	In	Yellowstone Park		
Tuesday,	"	21st.......	In	Yellowstone Park		
Wednesday,	"	22d........	In	Yellowstone Park		
Thursday,	"	23d........	Lv.	Yellowstone Park	P. M.	via N. P. R. R.
Friday,	"	24th......	Arr.	Butte.................	A. M.	via N. P. R. R.
Friday,	"	24th......	Lv.	Butte.................	P. M.	via N. P. R. R.
Saturday,	"	25th......	Arr.	Anaconda..........	A. M.	via N. P. R. R.
Saturday,	"	25th......	Lv.	Anaconda..........	P. M.	via N. P. R. R.
Sunday,	"	26th......	Arr.	Spokane............	A. M.	via N. P. R. R.
Monday,	"	27th......	In	Spokane............		
Tuesday,	"	28th......	In	Spokane............		
Wednesday,	"	29th......	In	Spokane............		
Thursday,	"	30th......	Lv.	Spokane............	A. M.	via N. P. R. R.
Thursday,	"	30th......	Arr.	Seattle..............	P. M.	via N. P. R. R.
Friday,	Oct.	1st.......	In	Seattle..............		
Saturday,	"	2d........	In	Seattle..............		
Sunday,	"	3d........	Lv.	Seattle..............	P. M.	
Monday,	"	4th......	Arr.	Tacoma.............	A. M.	
Monday,	"	4th......	Lv.	Tacoma.............	P. M.	
Tuesday,	"	5th......	Arr.	Portland......A. M.		via Regulator Line Steamer
Tuesday,	"	5th......	Lv.	Portland......A. M.		via Regulator Line Steamer
Wednesday,	"	6th......	Arr.	Salt Lake...........	P. M.	via O. S. Line
Thursday,	"	7th......	In	Salt Lake...........		
Friday,	"	8th......	In	Salt Lake...........		
Saturday,	"	9th......	In	Salt Lake...........		
Sunday,	"	10th......	Lv.	Salt Lake about 2.30	P. M.	via D. & R. G.
Monday,	"	11th......	Arr.	Pueblo.....about 1.30	P. M.	via D. & R. G.
Monday,	"	11th......	Lv.	Pueblo................	P. M.	via C. R. I. & P.
Wednesday,	"	13th......	Arr.	Chicago..............	A. M.	

On the way to St. Paul (reached at 9 p.m.) many interesting and beautiful views of the Mississippi were afforded; but nothing was seen of St. Paul or Minneapolis—the train proceeding without delay westward over the Northern Pacific railway.

The morning of Friday, Sept. 17, found the travelers in the State of North Dakota, Minnesota having been traversed during the night, and before the next morning they had proceeded far into Montana. About midnight they were joined at Billings, Mont., by a delegation from Denver, comprising President Brunton and a congenial party of members and guests.

Yellowstone Park.

On Saturday morning, Sept. 18, after breakfasting on the train at Gardiner, the terminus of the Yellowstone branch of the Northern Pacific, we were driven in six-horse coaches to Mammoth Hot Springs. Already before reaching Gardiner, sublime mountain scenery had been visible; and Sepulcher and Electric peaks, which tower above the railway station, fitting sentinels to the Park. A formal entrance is now furnished by a massive archway of lava blocks. The station at Gardiner is a picturesque structure of logs from the forests of Montana.

The drive to Mammoth Hot Springs, five miles long, ascends about 1,000 ft., to the altitude of 6,215 ft. To the writer, who first explored this region in 1871, before it was a National Park, or contained any roads or buildings whatever, and whose second visit was in 1888, the improvements in roads, hotels, etc., were a perpetual source of wonder and pleasure.

It is not necessary here to describe in detail the marvels and attractions of the Park. The beautifully illustrated railroad literature of the subject, to say nothing of travelers' descriptions and government reports, is full, graphic, satisfactory, and easily obtainable by all. This chronicle will be mainly confined to the experiences of our party.

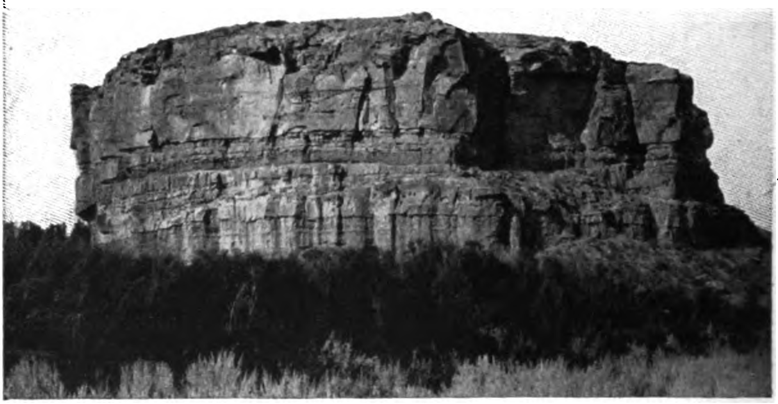
The remainder of Saturday was spent at Mammoth Hot Springs; and the succeeding days were occupied as follows:

Sunday, Sept. 19: A drive of 20 miles, beginning at 8.15 a.m., to the Norris Geyser Basin, where lunch was taken, and the drive was continued 20 miles further to the Lower Geyser Basin, where the night was spent at the Fountain Hotel. The day was fair. During the evening a Bible-talk was given by Dr. Raymond on the character of St. Paul and his attitude on the labor-question.

Monday, Sept. 20: A drive of nine miles, beginning at 9 a.m., to the Upper Geyser Basin, passing the Midway basin (where the Excelsior geyser, Prismatic lake, Turquoise spring, etc., are situated), and the Riverside and Lone Star geysers, etc. Lunch at the magnificent rustic Old Faithful Hotel (the finest log cabin in the world), and visits in the afternoon to Old Faithful, the Beehive, Giant, Grotto, Castle, Fan, Sawmill, etc.—the last four of which, in addition to Old Faithful (which offers its splendid spectacle hourly), were seen in eruption.

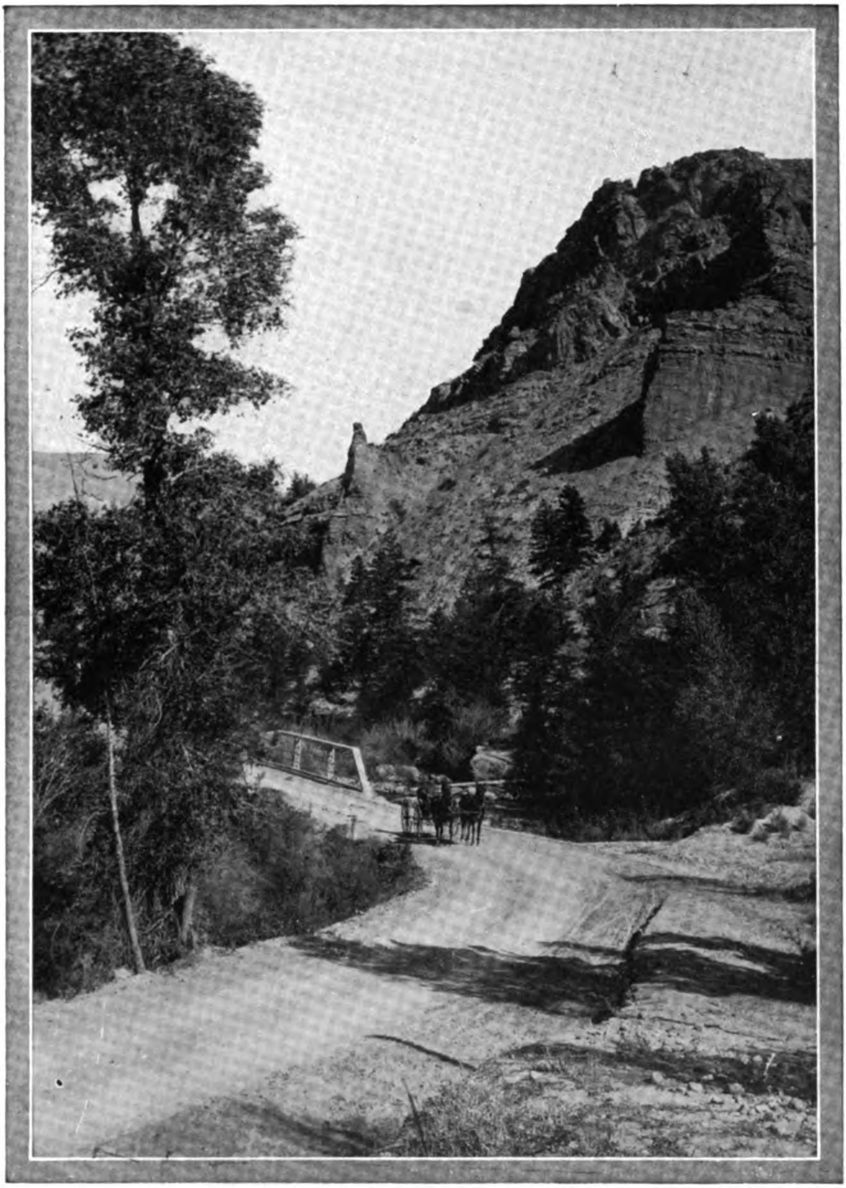
After nightfall, an electric search-light, turned upon Old Faithful during its majestic outburst, furnished prismatic effects of unearthly beauty and variety. The day was fair, but at bedtime it had begun to snow. During the evening, the company gathered around the great log fires (of which there were four in the massive stone chimney-stack), and corn was popped, during the eating of which the Secretary, upon request, told the story of his "internment" at Bologna, in 1860, as a military prisoner of the Garibaldian army in Italy, and the comical manner of his release.

Thursday, Sept. 21, the ground was lightly covered with



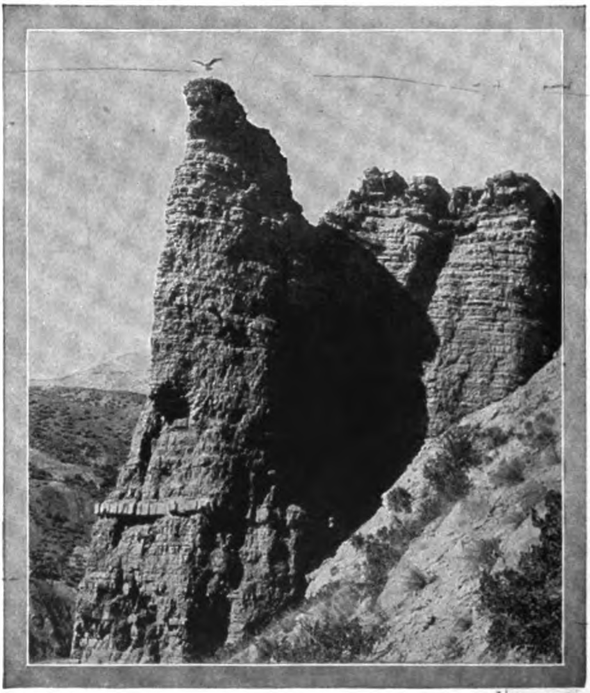
Pompey's Pillar, Yellowstone Valley, Mont.

snow, and snow-flurries occurred at intervals during most of the day. Yet the drive of 19 miles to the West Arm or "Thumb" of Yellowstone lake was not spoiled by the weather. The snow had laid the dust—no small matter, since the sprinkling of the roads at government expense had just ceased for the season—and the stately forests, the lovely Keppler cascade, and view of the white peaks of the Tetons from Shoshone point, furnished picturesque attractions. At the lake, after lunch, the party was divided—one-half proceeding to the Yellowstone Lake Hotel (16 miles) by coach, and the remainder making the trip on a small steamer to the same point. Those who went by land had the advantage, for a keen wind on the lake made

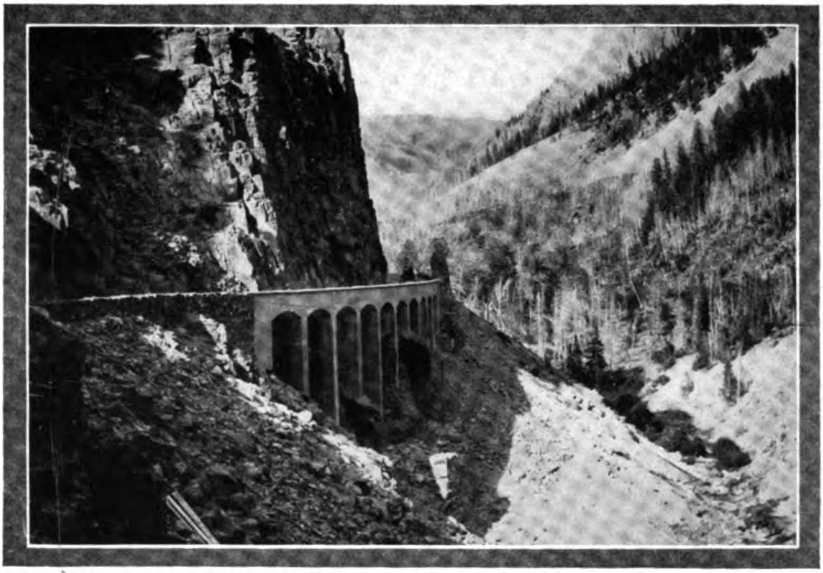


GARDINER CAÑON, YELLOWSTONE PARK.

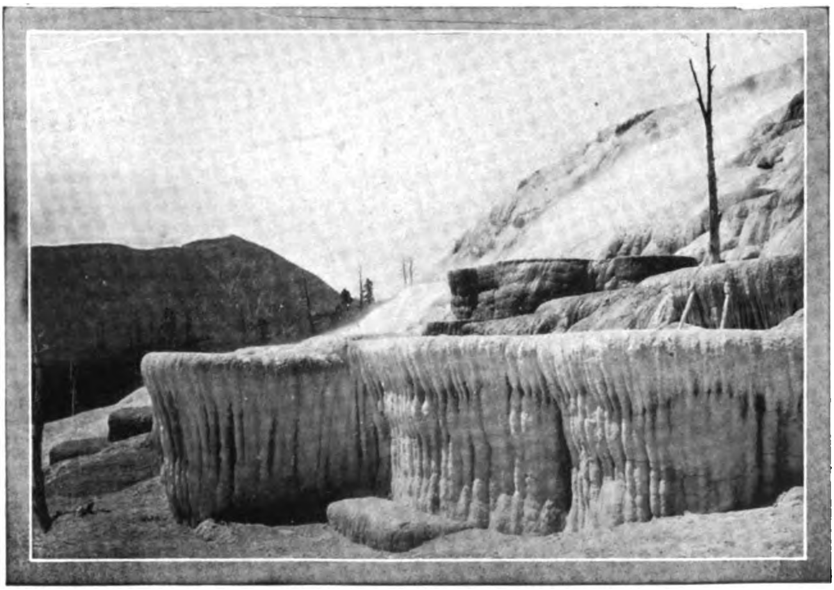
STAGES IN GARDINER CAÑON.



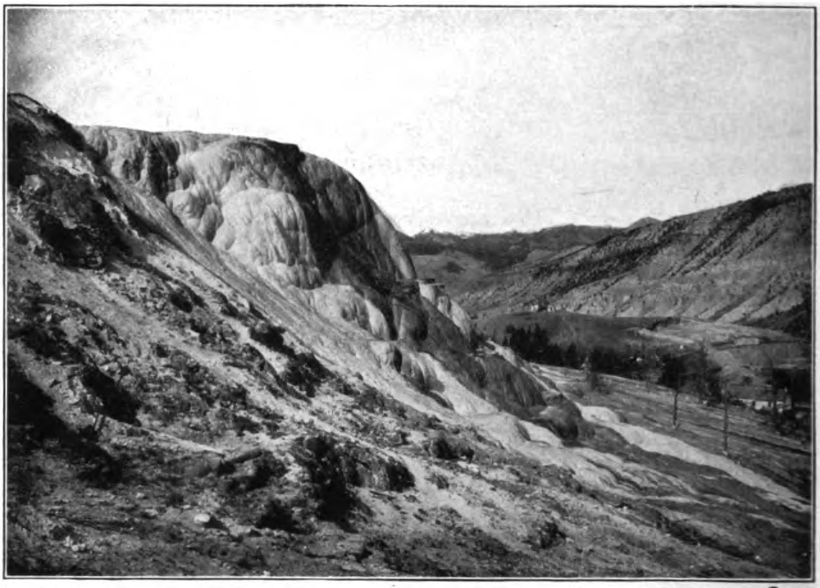
EAGLE NEST CRAG, GARDINER CAÑON.



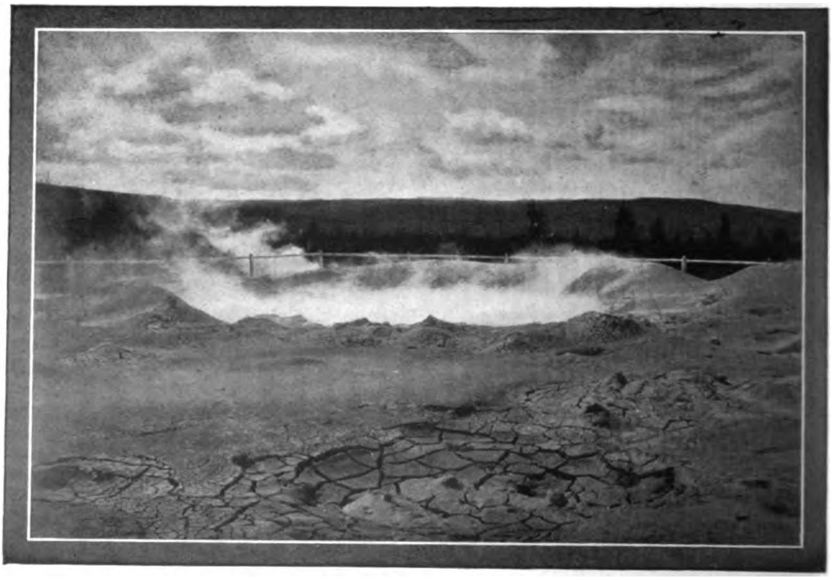
Golden Gate.



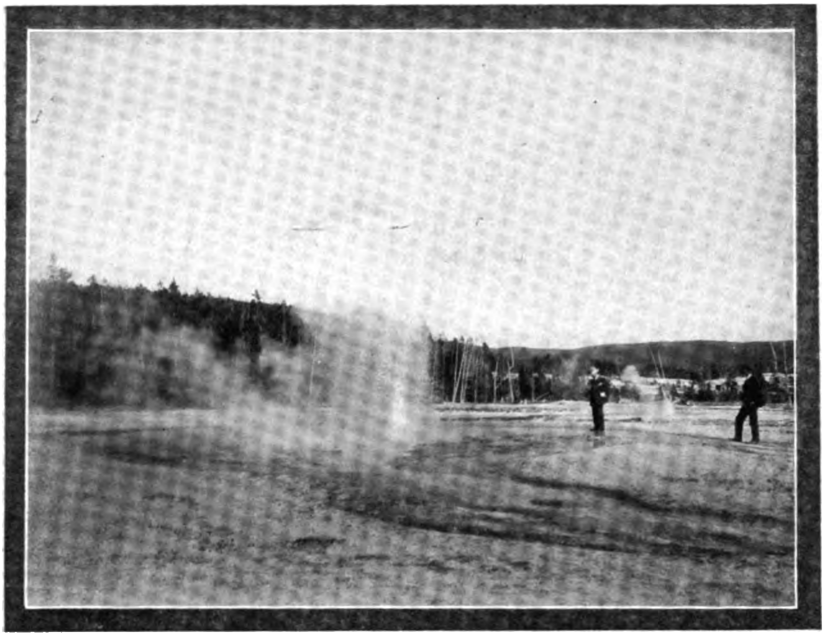
Pulpit Terrace, Mammoth Hot Springs.



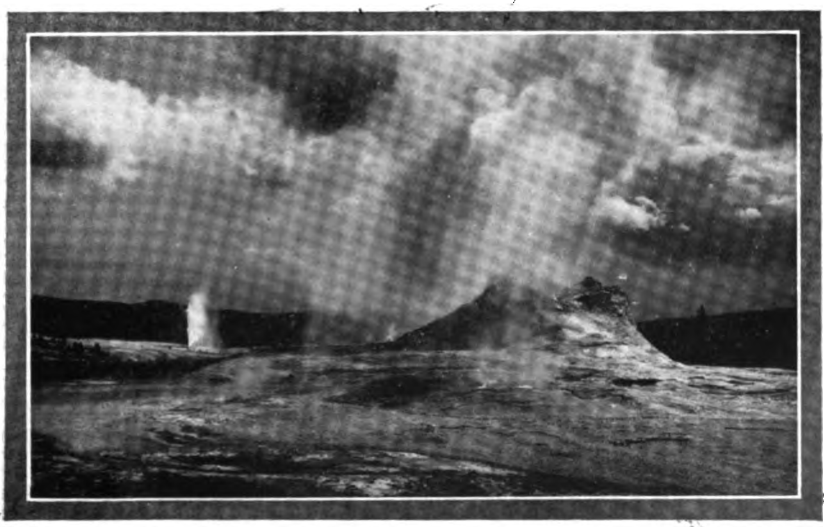
JUPITER TERRACE, MAMMOTH HOT SPRINGS.



MAMMOTH PAINT POTS, LOWER GEYSER BASIN.

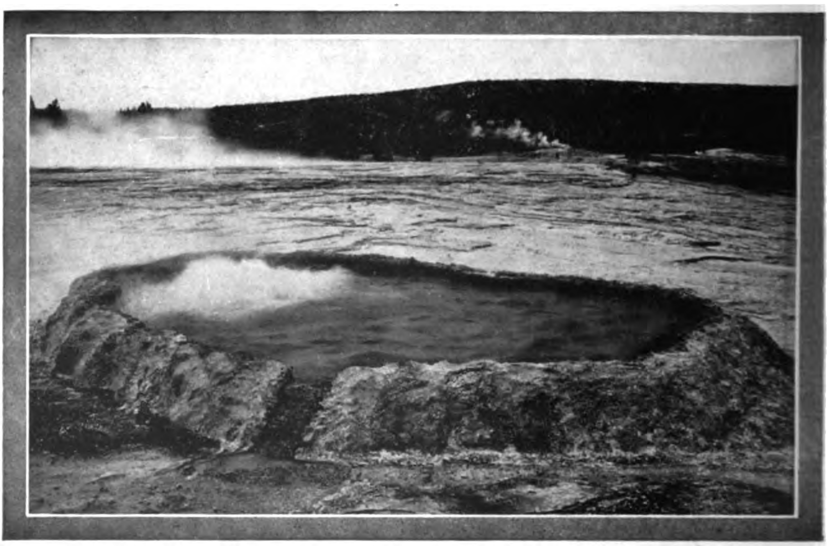


Sawmill Geyser, Upper Geyser Basin.



Castle, Beehive, and Old Faithful Geysers, Upper Geyser Basin.

Economic Geyser, Upper Geyser Basin.



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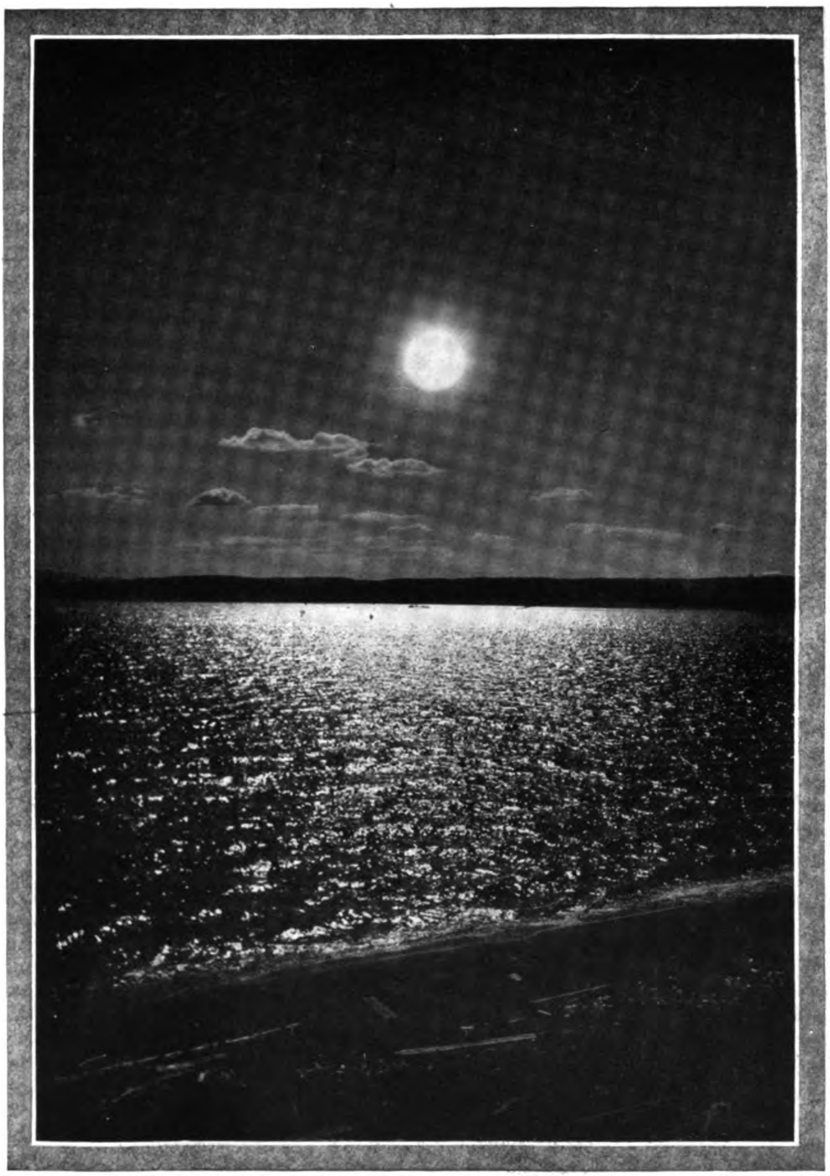
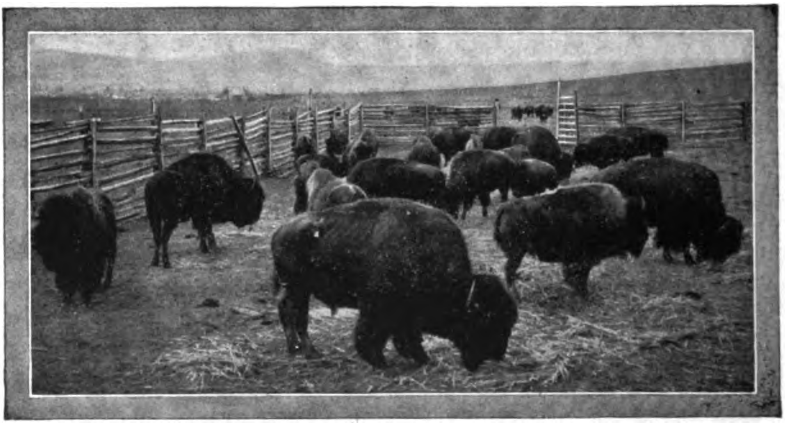
Tea Kettle Geyser, Upper Geyser Basin.



MOONLIGHT ON YELLOWSTONE LAKE.

YELLOWSTONE PARK DEER.

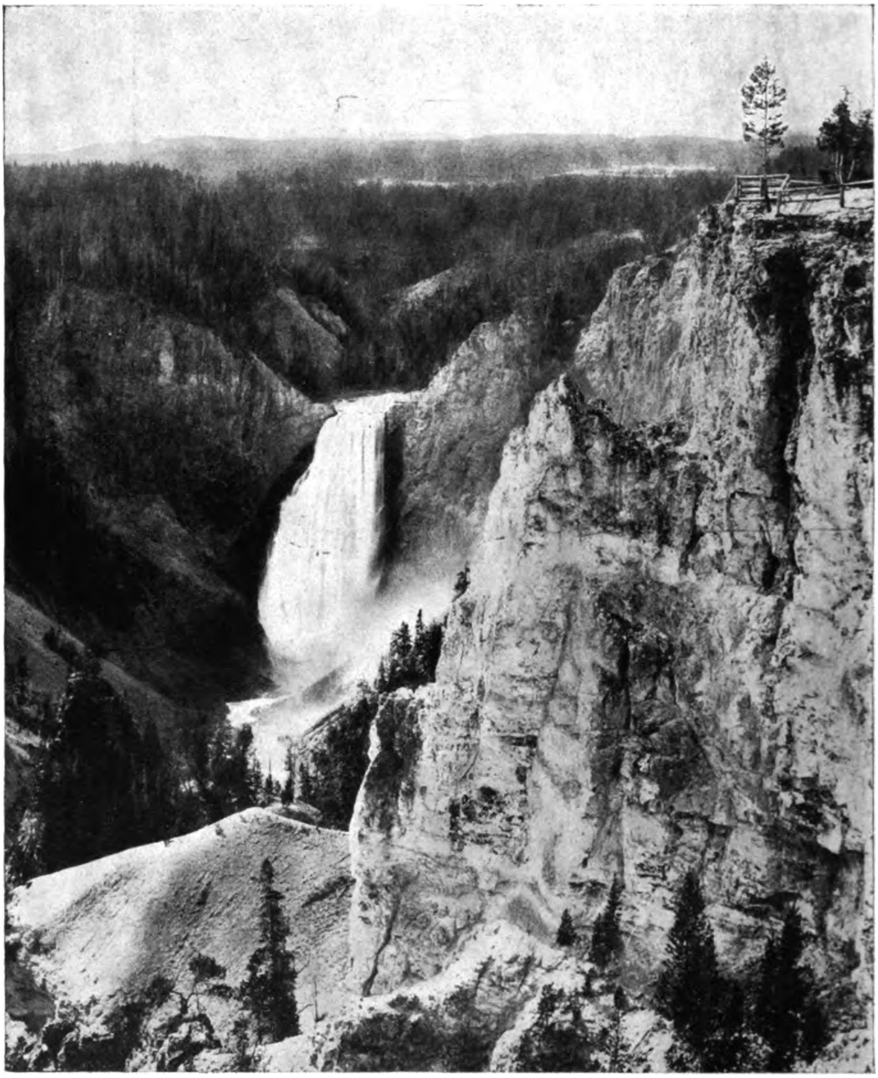
PARK BEARS—FORAGING FOR SUPPER.



SOME OF THE PARK BISON.

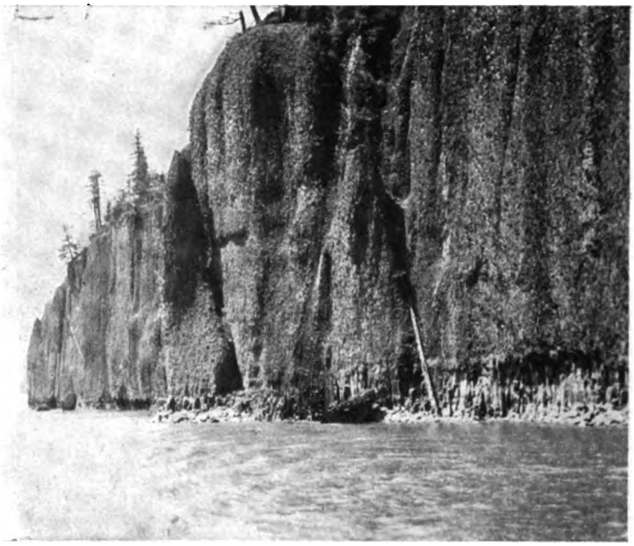
ANTELOPE IN PARK NEAR GARDINER.

NEW MELAN ARCH BRIDGE OVER YELLOWSTONE RIVER NEAR GRAND CAÑON.

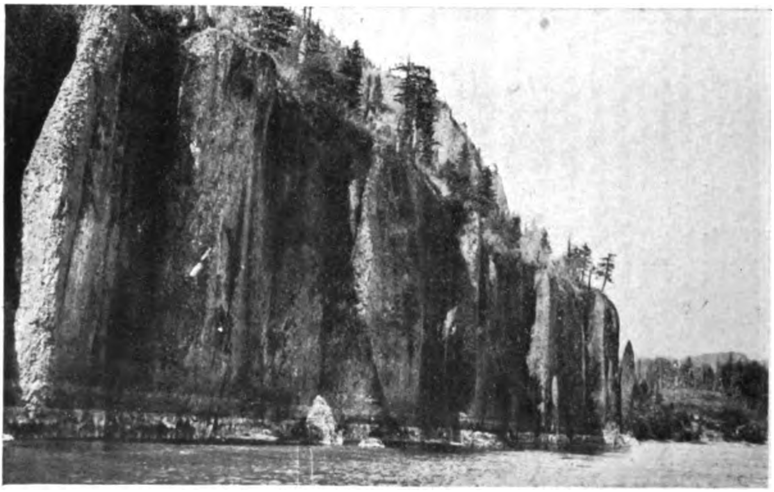


Grand Cañon and Lower Fall of the Yellowstone.

A Washington Forest.



Cape Horn, Columbia River, Wash.



Cape Horn, Columbia River, Wash.

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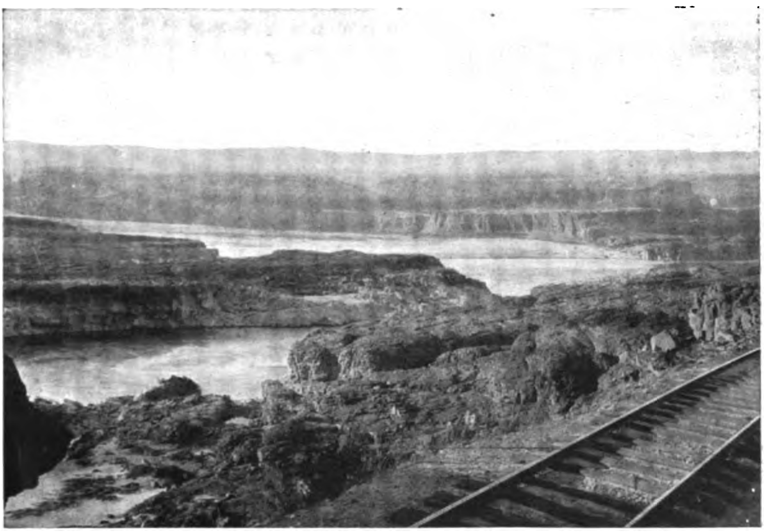
CASTLE ROCK, COLUMBIA RIVER, WASH.



DALLES OF THE COLUMBIA RIVER, WASH.

THE INSTITUTE PARTY AT TH FORESTRY BUILDING, A.-Y.-P. EXPOSITION, OCT. 1, 1909.

Top Row.

F. A. Thomson,	George W. Evans,	Mrs. George W. Evans,	Miss Louise Saunders,	Mrs. W. S. Pilling,
W. S. Pilling,	A. F. Blair,	D. O. Brooke,	Mrs. D. O. Brooke,	William Lilly,
Mrs. J. Labarth,	J. Labarth,	J. G. Perry,	A. T. Shurick.	

Second Row.

J. B. Adair,	Mrs. J. B. Adair,	Miss J. Saunders,	C. E. Magnusson,	Miss Emily Saunders,
Mrs. W. B. Saunders,	W. L. Saunders, Jr.,	Miss J. A. Glendenning,	Mrs. J. T. Haas,	J. T. Haas,
E. S. Hutchinson,	Charles W. Goodale,	Mrs. A. E. Vaughan,	Miss J. L. Douglas,	Miss Milnora Roberts,
A. Anderson.				

Third Row.

Mrs. C. E. Phœnix,	C. E. Phœnix,	Mrs. E. C. Knight,	E. C. Knight,	William Kent.

Fourth Row.

C. R. Claghorn,	Frank Caldwell,	Mrs. John Lilly,	F. H. Bostwick,	John Lilly,
Mrs. F. A. Hill,	Mrs. William Kent,	Mrs. E. J. McCaustland,	E. J. McCaustland,	Mrs. F. Keffer,
Frederic Keffer,	Mrs. C. M. Lewis,	L. P. Tolman,	M. K. Rodgers,	Mrs. George Jammes,
George Jammes,	Milton Moss.			

Fifth Row.

Milnor Roberts,	T. D. Jones,	C. M. Lewis,	Reiji Kanda,	W. S. Ayres,
Mrs. W. S. Ayres,	Mrs. J. W. Nesmith,	Mrs. F. L. MacFarland,	—— ——	—— ——
Mrs. H. S. Chamberlain,	H. S. Chamberlain,	A. R. Bellinger,	S. H. Pitkin,	S. H. Taylor,
M. J. Falkenberg.				

Sixth Row.

E. Jacobs,	C. M. Lewis,	Miss Cortwright,	W. H. Adams, Jr.,	Mrs. Van Deusen,
R. W. Raymond,	Mrs. William Kelly,	Mrs. J. G. Perry,	J. W. Nesmith,	W. J. Rankin, Jr.,
Charles Catlett,	—— ——	S. T. Wellman,	Percy Williams,	R. M. White.

Bottom Row.

W. H. Adams,	N. W. Emmons,	E. W. Parker,	J. Struthers,	D. W. Brunton,
W. L. Saunders,	Mrs. Frank A. Hill,	William Kelly,	Mrs. J. A. Mitchell,	J. A. Mitchell,
Mrs. M. H. Harrington,	M. H. Harrington	Mrs. A. W. Jenks,	C. F. Lee,	Master Rodgers.

it necessary to close the curtains on the steamer, and also roughened the water unpleasantly.

Dinner was taken at the Yellowstone Lake Hotel, a fine building in the colonial style, and in comfort and convenience perhaps the best hotel in the park. A lively game of base-ball, in which some of the young ladies participated with surprising skill, occupied the sunset hour, which offered a scene of quiet beauty in earth, wave, and sky, auguring well for the weather of the following day.

Wednesday, Sept. 22, contrary to these favorable indications, the weather was fitful and threatening, though not actually stormy. A chilly drive of 16 miles brought us to the Grand Canyon Hotel in time for lunch, after which, in wagons, on horseback, and, at intervals, on foot, the glories of the upper and lower falls and the glowing cañon below them were explored and enjoyed. Through the gathering clouds in the sky the sun broke occasionally in rudimental splendor, giving the needed last touch to the colors of the variegated cañon-walls.

In the evening Dr. Raymond gave an account of his visit to the geyser basins, the lake and the Yellowstone cañon in 1871, the year after their discovery by the Washburn party. In his opinion, there had been in 38 years an elimination in the activity of the geysers, though it was difficult to decide the question quantitatively, in view of the outbreak of new geysers in the interim. Some of those first observed, especially the Giant and the Giantess, had certainly diminished in power and frequency of eruption. But Old Faithful seemed to deserve its name fully.

Thursday, Sept. 23, was a perfect, balmy, golden day, making the homeward drive to lunch at Norris Basin, and thence to Mammoth Hot Springs, a continuous delight. By 4 p.m. we were once more comfortably settled in our waiting train at Gardiner.

Our trip of five and one-half days in the Park was unanimously declared to have been perfectly successful. The autumn weather was at no time so severe as to cause serious suffering. The pestiferous gnats of the Park were all gone; the roads were not dusty, and the bracing air by day, like the roaring chimney-fires in the evening, promoted companionship in activity and in rest, respectively. By the time the party re-

sumed its special train at Gardiner the relations of its members had passed from mutual courtesy to congenial friendship. No better way could have been devised to prepare a company of travelers for harmony and pleasure through a long tour.

Butte.

The train was side-tracked at Livingston, Mont., until 11.45 p.m., when it proceeded westward over the main line of the Northern Pacific, arriving in the morning of Friday, Sept. 24, at Butte. It was again a perfect day—real "Institute weather."

After breakfast we were taken in charge by the Local Committee, and treated to a most interesting trolley-ride through the main streets of the city and up to the top of the hill on which are grouped the principal mines. At the Leonard mine nearly all the party, including the ladies, were lowered in cages to the 1,200-ft. level, and enabled to see stopes, pumps, ore-bodies, horses used in hauling underground, etc. The consolidation of mining ownerships in Butte has resulted in great economies of drainage, transportation, etc., as well as of technical administration, and the removal of nearly all the smelters has wrought an amazing change in the atmosphere. Grass and flowers may now be seen in dooryards; and the town itself has become worthy of handsome buildings. The fine statue of Marcus Daly looks down the steep main street upon a scene as crowded as ever, but fairer and brighter than his eyes ever saw in Butte.

Luncheon was served in the stately and beautiful house of the Silver Bow Club, where Mr. Cornelius F. Kelly, acting as toastmaster, introduced Mayor Charles Nevin, who welcomed the Institute to the city. Addresses were also made on behalf of the hosts of the occasion by Messrs. Eugene Carroll, President of the Silver Bow Club, and William Scallon, former President of the Anaconda Copper Mining Co.; and on behalf of the Institute by President D. W. Brunton, Vice-President W. L. Saunders, and Secretary R. W. Raymond.

Later in the day the ladies of Butte entertained their feminine visitors with an automobile-ride and afternoon tea; and our evening reception at the Club, followed by dancing and supper, crowned the experiences of a brief but busy and fruit-

ful day. It would not be possible to pack more instruction and enjoyment into the same number of hours.

Anaconda.

During the night the special train was hauled to Anaconda, where Saturday, Sept. 25—another bright day—was spent. The party was conducted through the famous Washoe smelter, and the brick-making departments of the Anaconda Co.; luncheon was served at the Anaconda Co.'s Montana Hotel; and a visit was made to the County Fair. The significance of this exhibit was appreciated by those of the party who were familiar with the long conflict over the effect of the smelter-fumes upon agriculture, and the immense outlay of the Anaconda Co. for the purpose of removing this ground of complaint and litigation. The display of magnificent grains, fruits, and vegetables and fine cattle, grown within the area formerly thus afflicted, was a convincing object-lesson, so far as present conditions are concerned.

The following description of the Washoe smelter is condensed from an account prepared by members of the technical staff of the Anaconda Copper Mining Co.:

This smelter is about two miles east of the city of Anaconda, on a site including about 240 acres of hillside and plain.

The ore (10,000 tons daily) comes principally from Butte, a distance of 28 miles, in trains of 50-ton bottom-dump cars.

Concentrator Bins.—These are divided as follows: Eight second-class ore-storage bins for 1,250 tons each; thirteen sample-ore bins for 200 tons each, and one coal-storage bin of 2,500 tons capacity.

Sampling-Mill.—This is of wood throughout, 40 by 60 ft., and 6 stories high. It is in 2 sections, each having a capacity of 1,800 tons in 24 hours.

The ore to be sampled is taken from the sample bins by an electric locomotive, dropped into bins feeding the crushers, and, when discharged from the crushers, elevated to the top of the building by bucket-elevators. Brunton automatic samplers are used, cutting out one-fifth of the stream each time and discarding four-fifths—the final result being after 4 such divisions 3.2 pounds for each ton of ore sampled.

Concentrator Boiler-House.—This is a brick-and-steel building, which furnishes steam to the concentrator power-house. It has ten 300-h.p. Stirling boilers, necessary feed-pumps, etc., and an independent steel stack, 14 ft. in diameter and 160 ft. high.

Concentrator Power-House.—This concentrator power-house, a brick and steel building, has two 4-cylinder triple-expansion condensing engines, each capable of developing 3,300 i.h.p. at 200-lb. steam-pressure. The economical i.h.p. is 1,750, with 150-lb. steam-pressure, at which they are run. There is also a 4-cylinder triple-expansion Fraser & Chalmers 1,150 i.h.p. engine. Since electric

installations have been made to use the power furnished by the Missouri River Power Company, the above engines are kept in reserve.

Electric Power.—This is furnished by the hydro-electric plant of the United Missouri River Power Company, near Helena, Montana, and the hydro-electric plant of the Anaconda Copper Mining Company at Flint Creek Falls, 22 miles west of Anaconda. The current from the Missouri River Power Company's plant is transmitted at 60,000 volts; that from Flint Creek Falls at 22,000. Both are stepped down to 2,200 volts at the Anaconda sub-station, and distributed for use in city and works lighting, street-car system, small power circuits in the city, and all electric power required in the works for motor and crane service. About 5,000 electrical horse-power is used.

Concentrator.—This consists of two steel and wood buildings, each in size 255 by 350 ft., and containing four complete sections, or eight in all. Each section is equipped with the same kind of machinery, and will handle 1,100 tons of ore in 24 hours. The equipment of one section is as follows: One 12 by 24 in. Blake crusher; 2 Blake crushers, 15 by 5 in.; 6 coarse-concentrate Harz jigs; 1 set of 15 by 42 in. coarse rolls; 1 set of 15 by 42 in. fine rolls; 36 fine jigs (Evans patent); 1 set of 15 by 42 in. middlings rolls; 12 middlings jigs (Evans patent); 3 Huntington mills, 6-ft.; 12 fine finishing-jigs (Evans patent); 33 Wilfley tables; besides a large number of classifiers, settling-tanks, dewatering tanks, elevators, and trommels. A system of launders carries the concentrates to the settling-tanks at the foot of the building.

The tank house, a 70 by 670 ft. frame building, contains nine settling-tanks for each section. These are 19 by 19 by 15 ft. high, each having a capacity of 420 tons of concentrates. It is the practice at this plant to weigh and sample the material entering and leaving every building, so that an accurate check is kept on the work of each department.

Slime Ponds.—These are in the valley below the works. There are six ponds of different sizes, but averaging about 300 by 630 ft., and the slime water, containing the greater metal values coming from the concentrator, goes to these ponds for settlement. When a pond is full, the water is diverted to an empty pond, and the former is drained. The pond is excavated by a Lidgerwood traveling cableway, with a bucket capacity of 5 tons. The slime is piled outside, allowed to drain and dry, and taken by cableway and dropped into a hopper on trucks, from which it runs into the railroad cars beneath and is taken to storage-bins for use at the briquette plant. The slime water containing the least values is used for condensing purposes and for sluicing slag at the reverberatories and blast-furnaces.

Roaster Building.—This 96 by 412 ft. building is of steel throughout. It contains 64 McDougal calcining furnaces of the Evans-Klepetko type, each 18 ft. high, each with six hearths, 16 ft. in diameter; revolving water-cooled shafts and arms driven by gearing from below. No fuel is used other than the sulphur in the concentrates, the burning of which furnishes sufficient heat to do the calcining, except on occasions when the furnace is not hot enough to ignite the sulphur, at which times fine coal is fed. The gases are taken through brick flues into large brick and steel dust-chambers, where a large proportion of the flue dust is settled. Each furnace is capable of handling 45 tons of material in 24 hours.

Reverberatory Buildings.—These are two steel buildings, each 183 by 225 ft., and each containing four coal-fired furnaces, the hearth dimensions of which are 19 ft. in width and from 102 ft. to 116 ft. in length, with a grate area of 8 by 16 ft. and a smelting capacity of 300 tons per 24 hours on natural draft.

The fuel used is Diamondville coal, shipped from the mines in Wyoming, owned by the Washoe Copper Co.

The flame, after leaving the furnace, passes through two 375-h.p. Stirling boilers, in tandem, which reduce the temperature of gases going to the main flue to about 600° F. By this means 600 b.h.p. are obtained from each furnace from the waste heat. The ashes and partially burned coal that drop from the fire-box fall into a stream of water which carries them over a grizzly, the larger pieces of ash going to the slag-sluice ; the smaller, containing coke and unburnt coal, being sluiced to the coke-jigging plant, where the coke and coal are jigged out and the ashes sent to the dump. The coal and coke recovered are elevated to a bin, from which they are taken to the briquette plant by the local tramming system and there become a constituent part of the briquettes, thus obtaining a fuel value from an otherwise waste product. By this means 10 per cent. of the reverberatory fuel is recovered.

The matte is kept down some distance below the skimming-plate, making it impossible to "pull" out any of the matte, thus avoiding explosions. The matte is tapped from the side of the furnace, through copper tap-hole plates, and run through cast-iron launders, lined with siliceous material, to hot metal ladles of 10-ton capacity, and taken by the local tramway to the converters. The brick used in these reverberatories are manufactured by the brick department of the Anaconda Copper Mining Co., which is located in Anaconda. They have given the greatest satisfaction.

Briquette Plant.—This is a frame building, 55 by 192 ft., containing four Chambers Brothers end-cut, auger-type brick machines—each machine having a capacity of 700 tons of briquettes in 24 hours. The briquettes are made of fine concentrates, fine first-class ore, pond-slime, and fine coke from the reverberatory ashes. These materials are conveyed from the storage-bins by belt-conveyors to a pug-mill, from which they discharge into the brick machine proper, where they are further mixed and forced through a former by the auger of the machine in a continuous bar. This bar is cut into briquettes, weighing about 10 lb., by a revolving cutter peculiar to this type of machine. The briquettes are conveyed by a series of belt-conveyors to storage-hoppers, from which they are loaded into blast-furnace charge cars as part of the charge.

Blast-Furnace Building.—This steel building, 82 by 269 ft., contains three furnaces—two of which are 51 ft. long and the other 87 ft. long—having a width of 56 in. at the tuyeres. The smaller furnaces have a capacity of 1,600 tons, the larger 3,000 tons of charge in 24 hours.

The bottom of the center of the furnace is of silica brick, laid on water-cooled cast-iron plates, mounted on cast-iron columns, and has a gradual slope to each discharge-spout. The 87-ft. furnace has three discharge-spouts and three settlers, but is otherwise built in the same manner as the 51-ft. furnace. The 51-ft. furnace has 88 tuyeres, and the 87-ft. furnace has 150 tuyeres, all 4 in. in diameter.

The type of furnace used is the Mathewson patent blast-furnace. It is very successful in operation and much easier to handle than the old-style furnaces. Its advantages are: (1) larger hearth-area, with but two ends to bind and hold the crusts (any crusts forming on the sides can be readily got rid of by allowing the furnace to run down, the crust either dropping or being readily barred); (2) smaller radiating surface for the same hearth-area than the smaller furnaces; (3) smaller coke-consumption ; (4) flexibility as a unit—any part of the furnace being handled or repaired without shutting down the whole. The entire end of one furnace has been shut down, jackets changed, furnace cleaned out, and operations resumed—and during all this time (extending over ten days to two weeks) the other half of the furnace was in operation. The 51-ft. furnace has three 7-ft. 45-degree unlined steel flues ; the 87-ft. has five. All the flues discharge into a large

brick and steel dust-chamber of the type adopted for the entire plant. The dust-chamber is connected by a large flue to the main flue.

The furnaces are charged from both sides, the doors being raised by compressed air. A "charge train" consists of eighteen cars, which receive the weighed quantities of the various materials from the storage-bins adjacent to the building. These bins are in three rows and are built of wood. Each row is 28 ft. wide, 786 ft. in length, and 20 ft. deep, and divided into a series of bins of various sizes as required by the volume of material handled. They act as storage for first-class ore, coarse concentrates, lime rock, coal for reverberatory and power-house, slime for briquetting, converter lining, etc., and are filled from the B., A. & P. tracks on top of the bins. All gates in these bins that are used constantly are operated by compressed air. The charge train first takes its quota of lime rock, then slag, then ore, then coarse concentrates—then goes to the briquetting plant, where it receives its quota of briquettes. Two charge cars constitute a charge, the weight of which varies from 8,400 to 12,000 lb., according to its composition. The train when loaded is hauled into the blast-furnace building, where the cars are dumped by compressed-air lifts.

The slag and matte flow from the furnace through the discharge spouts into the 16-ft. settlers, previously mentioned. The settlers are circular and made of half-inch steel plates lined with 12 in. silica brick. The slag overflows and is granulated by slime water from the concentrator, and is carried off in launders lined with cast-iron to the dump. The matte is tapped from the settler into the 10-ton hot metal ladles of the local tramming system and taken to the converter plant while still molten.

Converter Building.—This building, including the lining plant, casting-furnaces, and converters, is 176 by 516 ft., and is constructed of steel, except the crushing and mixing department of the converter lining plant, which is of wood. There are 13 stands for horizontal barrel converters, 8 ft. in diameter and 12 ft. 6 in. long, operated hydraulically. There are three 60-ton electric traveling cranes—two for the handling of converters, etc.; the other for slag and copper. There are also two 15-ton electric cranes in the casting department for general use. The converter is filled in a nearly vertical position with the air blast on. Seven tons of matte per charge is used whether the converter is freshly lined or not, and the charge, as a rule, is finished to blister copper in the same converter.

The slag from the converters is carried in steel ladles to the casting-machines, from which it is loaded into railroad cars and transferred to the storage-bins for blast-furnace use.

The blister copper is poured into a sheet steel clay-lined ladle, and taken by the crane to a hydraulic cradle, from which it is poured into the casting-furnace.

The converters are lined in the main converter building, but the lining material is prepared in a separate 40 by 62 ft. building. The lining material is highly siliceous ore, having gold and silver values, with pond-slime as a "binder."

Refining and Casting.—Of three casting-furnaces, two of 110 tons and one of 140 tons, two are in operation all the time. The casting machine is a platform conveyor with the molds attached and operated hydraulically. It is capable of casting 25 tons per hour.

The copper is chilled by a spray and when "set" is dumped from the mold onto a platform conveyor, operating through a tank of water, then trucked to scales, weighed and shipped to Eastern refineries.

Smelter Power-House.—This 85 by 526 ft. brick and steel building contains the engines, compressors, blowers and auxiliaries necessary to produce the various pressures of air required. There are six Connersville and two Roots blowers,

direct-connected to Corliss engines, each having a capacity of 300 cubic ft. per rev., compressing to 40 ounces 360,000,000 cubic feet of free air in 24 hours for blast-furnace use; also, seven 16-pound air compressors, compressing about 60,000,000 cubic feet of free air in 24 hours for converter use; three 90-pound air compressors for general use, such as shop tools, air gates, raising blast-furnace doors, dumping blast-furnace charge cars, tamping converters, etc.; four 900-pound air compressors for the air locomotives of the local tramming system; 4 hydraulic pressure pumps and accumulators, pumping water up to 360 lb. for use in hydraulic apparatus at converter plant, and all the necessary auxiliaries, such as gravity condensers, dry vacuum pumps, feed-water heaters, boiler feed-pumps, etc. The steam for these engines is supplied from the waste heat boilers at the reverberatory building and from twelve Stirling boilers in a boiler-room bricked off from the power-house. Many of these engines are kept as reserve units since the introduction of electric power.

Electric Power-Plant.—The blower plant is a brick and steel building, 46 by 156 feet, and contains four Roots blowers belted to 600-h.p. induction motors, each having a capacity of 315 cu. ft. per rev., compressing capacity 220,000,000 cu. ft. of free air to 40 ounces pressure in 24 hours, for blast-furnace use.

Local Tramming System.—The equipment consists of seventeen of H. K. Porter & Co.'s air locomotives, weighing from 12½ to 22 tons each, and 240 cars of various kinds. The locomotives operate with 900-pound air-pressure in the receivers, which is reduced by means of reducing valves to 150 pounds for use in the cylinders.

There are 12.48 miles of standard gauge tracks in the works.

This department handles about 13,000 tons of material in 24 hours. The charging stations are placed at convenient points throughout the works and are fed by an extensive piping system. The locomotives are charged about every 20 minutes—this varies according to the nature of the work. It takes about two minutes to charge.

Flues and Stacks.—The flue arrangement is noted for its immense size. The three principal flues, viz., the blast, roaster and reverberatory, are 20 ft. wide and 15 ft. high and of brick and steel construction. The converter flue consists of two 7 by 7-ft. flues. The blaster, roaster and converter flues connect with their respective dust chambers; the reverberatory flue with the furnaces direct. The flues are of the following lengths:

	Feet.
Blast,	1,653
Roaster,	488
Converter,	703
Reverberatory,	1,253

These flues merge into one main flue.

For the first 1,234 ft. this flue is 60 ft. wide; side walls 20 ft. high; the bottom being excavated at an angle of 30 degrees from the horizontal. The roof is of I-beam and brick arch construction. The remaining distance to the track is 995 ft. of 120-ft. flue.

This portion of the flue has a roof of No. 9 sheet steel. The stack is 300 ft. high, with an inside diameter of 30 feet. The top of this stack is 932 ft. above the surrounding valley.

The flue dust is drawn off through hoppers, spaced every 10 ft. in the tunnel, into cars operated by the gravity system from a set of drums placed immediately behind the stack. When the cars are loaded they are sent to the lower end of the

main flue and elevated to the adjacent flue dust bins. There are two sets of bins here, one for the flue dust containing the desired percentage of arsenic for the arsenic plant, the other for flue dust to go directly to the reverberatories.

Arsenic Plant.—The flue dust intended for the arsenic plant is conveyed from the bins by inclined revolving pipes into the feed hoppers of two Brunton revolving hearth roasting furnaces. The arsenic fumes from these are conducted through 240 ft. of zigzag flue, cooling off the gases from which sublimes the arsenic giving a product of about 90 per cent. As_2O_3. When a sufficient quantity is produced, the roasting furnaces are shut down, and this product handled by wheel-barrows to a small reverberatory refining-furnace, fired with coke, and re-sublimed in a similar zigzag flue, thus getting a product of 99.80 per cent. As_2O_3, which is then ground and barreled for shipment.

All residues from the roasting furnaces are transported by the local tramming system to the reverberatory furnaces for smelting.

Change-House.—For the convenience of the men there is provided a brick change-house, 55 by 97 ft., containing 1,528 lockers, steam-heated, and ventilated by an independent system of flues to the outside air. Hot and cold water are provided, as well as shower baths. In this building there is an emergency hospital, with a trained nurse in attendance, who gives first aid to the injured and attends to the minor cases of injury. For serious cases a modern ambulance is available to take the patients to the hospital in the city.

Other Buildings.—The foundry and various shops, laboratory and testing departments, general office building, telegraph and telephone, engineering, photographic and electrical departments, need not be particularly described. They are extensive, complete, and up-to-date.

Water System.—The storage system for winter supply consists of two lakes—Storm lake and Silver lake. These are natural lakes, supplemented by dams to increase the volume. Storm lake receives its water from its natural drainage area; Silver lake is filled from its drainage area as well as receiving its greatest supply from the overflow, in the spring, of two lakes known as Twin lakes.

In the winter time a certain amount of water is pumped out of Silver lake storage to supplement the water coming from the creeks below. This lake is 15 miles west of Anaconda, at an altitude of 6,480 ft., or about 1,100 ft. higher than Anaconda. The water from both creeks and lakes is gathered by a dam about four miles west of the city and diverted into a 5 by 7-ft. flume, seven miles long, that carries it to the works.

Magnitude of Operations.—Some idea of this may be obtained from the following figures: Amount of ore that can be treated in 24 hours, 10,000 tons; lime rock from adjacent quarries, 2,300 tons; coke used, 550 tons; coal for reverberatory use, 500 tons; coal for power, 50 tons; water, per minute, 35,000 gal.; men employed in and around Anaconda, 3,000; monthly payroll, $300,000.

The Cœur d'Alène.

Supper was taken at home, *i. e.*, on the train, which was hauled during the night to Missoula, Mont., and thence to Mullan, in the Cœur d'Alène district of Idaho. The day, Sept. 26, was fair like its predecessors, and gave full effect to the beauty of the Cœur d'Alène scenery. The mill and Morning (silver-lead) mine of the Federal Mining & Smelting Co.

were here visited—the mine being reached by electric railway and a tunnel 2 miles long, and 2,400 ft. below the surface at its face. In the afternoon the Hecla silver-lead mine at Burke, near Wallace, was visited under the guidance of Mr. James F. McCarthy, the manager. The large direct-connected electric mine-hoist at this mine was studied and admired.

On Monday, Sept. 27—still fair!—the train "dropped down" to Wardner, where the Bunker Hill and Sullivan mine and mills were inspected, under the guidance of Mr. Stanley Easton, the manager, and the ladies enjoyed a trip over the "high line" to the Federal mine. In the afternoon our journey was continued to Harrison, on the shore of the lovely Cœur d'Alène lake. At this point we were transferred to one of the lake steamers of the Red Collar Line, and conveyed to the town of Cœur d'Alène. The special train, meanwhile, proceeded to Spokane over the "short line" of the Oregon Railroad & Navigation Co. This gave us an opportunity to gain a conception of the scenery of the region which ordinary transcontinental passengers do not enjoy. On the Northern Pacific, indeed, one gets a glimpse of Pend Oreille, but no hint of the splendor of that chain of lakes, including Cœur d'Alène to the south and the Kootenai region to the north, bordered by forest-clad hills or lofty mountains, receiving a thousand leaping streams, and furnishing at once a highway for the trader and a paradise for the tourist and the sportsman.

From Cœur d'Alène city the electric railway of the Spokane & Inland Empire Railroad Co. conveyed the party to Post Falls, where a stop was made to inspect, by the courtesy of Mr. D. L. Huntington, general manager, the new dam and power-plant of the Washington Water Power Co., which supplies power by electric transmission to nearly all the Cœur d'Alène mines as well as to Spokane. The train passed also through the irrigated district of the Spokane valley, the fertility of which afforded an indication of one of the elements of the prosperity of that city.

Spokane was reached an hour and a half behind schedule-time; and it became necessary, in consequence, to abandon the opening session of the meeting of the Institute, which was to have taken place that evening.

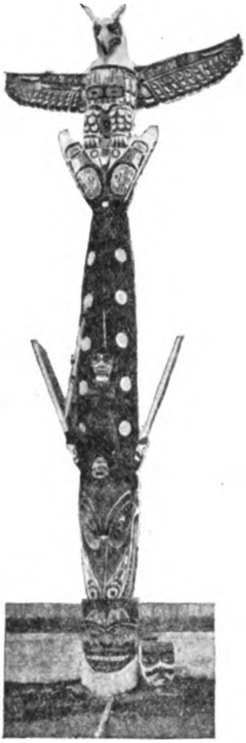
Spokane.

The proceedings of the meeting are reported elsewhere. Our long record of fine weather was broken on Tuesday, Sept. 28, by showers, which were, however, not heavy enough to spoil the enjoyment of the day. The forenoon was spent at the open-air reception and address of President Taft, for which the Institute party was provided with seats on the grand stand. Already on the night of the 29th the city had been brilliantly illuminated in honor of the President's coming, and the day-parade which he reviewed was a striking one. More impressive still was the immense throng of citizens which listened to his grave and earnest address. Some of the newspaper men, timing, watch in hand, the duration of the cheering with which he was greeted, and comparing it with the periods of "Bedlam broke loose" which have been made the fashion in other political meetings, were silly enough to telegraph to Eastern journals that Mr. Taft was received with noticeably little enthusiasm. We, who witnessed the scene, thought there was all the enthusiasm that good manners permitted, and congratulated ourselves and our friends of Spokane that America could still produce a vast audience of citizens who knew how to compliment a statesman by listening eagerly to his words, instead of yelling at him. There was frequent applause; but it came at the right points and was the expression of intelligent approbation. No doubt this admirable behavior was partly due to Mr. Taft's extraordinary clearness of delivery. In so vast a multitude people often cheer because they cannot hear. But Mr. Taft's voice carried his every word to every one of his many thousand listeners. Even we, who were behind him on the grand stand, heard everything as well as those who stood massed before him. To the remotest outskirts of the crowd no one departed, and every intent, upturned face was the face of one who heard and understood.

During the afternoon the ladies were entertained with an automobile-ride, and in the evening there was a dance. As an indication of the liberal dimensions and accommodations of the handsome Masonic Temple, it may be noted that during the evening the session of the Institute was held in one of its splendid halls, the dance in another, and a Masonic meeting in a third.

On Wednesday, Sept. 29, the showery weather continued. In the afternoon, however, various visits were made to the mineral exhibits, the Spokane tin-mines, the city power-plants, etc. And in the evening a banquet took place at the "Hall of the Doges," a chamber of the famous "Davenport restaurant," splendidly decorated in the Venetian style—though not at all like the hall for which it is named. Mr. C. M. Fassett presided, and addresses were made to the toasts: "Our Guests" (Hon. George Turner, of Spokane); "The Institute" (President D W. Brunton, of Denver); "Our Canadian Brothers" (Thomas Kiddie, of Northport, Wash., President of the Western Branch of the Canadian Mining Institute); "The Engineer Himself" (Vice-President W. L. Saunders, of New York); "The Mining School" (Prof. Francis A. Thomson, Washington State College); "Our Trip Experiences" (R. W. Raymond, of New York); and "The Ladies" (J. C. Ralston, of Spokane).

The impressions brought away from Spokane by the members of the party included not only its evident and growing prosperity, but also its peculiar attractions as a city of beautiful homes, inhabited by cordially hospitable people.



TOTEM POLE, ALERT BAY, ALASKA.

Seattle.

Thursday, Sept. 30, was a delightful day. The train left Spokane under a clear sky, at 7.30 a.m., by the Northern Pacific, and traversed the fertile region known as "The Inland Empire," including the magnificent Yakima valley, the reservation of the prosperous Yakima Indians, and many other sections in which irrigation has made the sage-plains to blossom as the rose and to bear incredible harvests of alfalfa, grain, and fruit. Climbing gradually among the picturesque Cascades, we passed the summit through the Stampede tunnel, and slid swiftly down the true Pacific slope through verdant forests and fields, reaching Seattle at 9.30 p.m. A portion of the party went to the new and admirable Hotel Washington, and the rest remained in their comfortable quarters on the train.

Here we encountered again the Presidential excursion of Mr. Taft, who had reached the place by a different route and was lodging at the hotel.

Friday, Oct. 1, was a gray day; but no rain fell until evening. In the forenoon the "re-grading" operations, in which some of the high hills are removed by hydraulic mining, to make way for business streets, were inspected with wonder. It is said that during the last year one-seventh as much excavation has been done for this purpose in Seattle as was performed on the Panama Canal!

A trolley-ride was then taken through the business and residential sections of the city. The site of Seattle, lying between Lake Washington and Puget sound, and penetrated from both sides by bays, is an ideal one for a great city, giving as it does ample port-facilities and, at the same time, room for parks and residences separated from the densely-built business districts.

The trolley-tour of many miles terminated at the gate of the Alaska-Yukon-Pacific Exposition.

Here the Institute party was received in the stately New York building, where luncheon was served, and an address of welcome was delivered by Mr. Samuel P. Weston, of the Department of Ceremonies. After a suitable reply by the President and the Secretary of the Institute, the afternoon and evening were spent in viewing the Exposition.

It is not necessary to describe here the attractions of this Exposition. Smaller than several of its famous predecessors, it was on that account more easy to comprehend and enjoy in a couple of days; and certainly it contained many beautiful, interesting and impressive exhibits, apart from the general fascination of its effective architectural plan and arrangement. Flowers by day and countless electric lights by night gave it a bewildering loveliness. In front of the great Forestry building of the State of Washington, the columns of which were the trunks of mighty trees, the party was photographed—and the result is shown in the half-tone engraving given on p. 1084. It was considered an extraordinarily successful out-door picture, because everybody in it could at least recognize himself!

Saturday, Oct. 2, was again fair, the rain having spent itself in the night. The time was occupied with trolley- and automobile-rides, and further study of the Exposition.

Sunday, Oct. 3, was a day of rest. A quiet and pleasant reception with afternoon tea was given at the residence of Prof. Milnor Roberts, of the Washington State University.

Tacoma.

On Monday, Oct. 4, the special train proceeded to Tacoma, where, under the guidance of the Local Committee, the city and suburbs were traversed by trolley—the most effective way ever invented for showing a town to strangers, comprehensively and satisfactorily, in a brief time. On this occasion the weather was fine, and many clear views were obtained of Mt. Tacoma and the range over which it towered on the horizon. (It is called Mt. Rainier on the government maps, and in the city of Seattle; but the Tacoma people stand stoutly for the old Indian name, and give their good reasons by the column or the hour. In this account, without intention of taking sides in the con-

troversy, the latter name is adopted, for the simple reason that when we were at Seattle, the air was too hazy to permit Mt. Rainier to be seen; consequently, the only peak we saw was viewed from Tacoma, and was emphatically and repeatedly called Tacoma by the numerous persons who pointed it out to us, and who looked as if they ought to know!)

Tacoma, like Seattle, has a beautiful and convenient site. Its long, deep-water harbor-front, its heights for residences, and its magnificent wild forest-park, are strong attractions. But its commanding commercial advantage seems to be a large level area of low ground, which the great transcontinental railroads have secured for their terminal facilities. Of these "flats," until recently disregarded as worthless, the Tacomans are now justly proud, seeing in them the prophecy of swift-coming greatness for their city.

The Institute party visited one of the greatest sawmills in the world—the mill of the St. Paul & Tacoma Lumber Co.—and followed with interest the transformation of huge pine logs to planks, boards, and shingles.

Luncheon was served at the Commercial Club, and the party afterwards visited the Tacoma smelter, on the way to which opportunity was given for a brief inspection of the new Tacoma high-school building (originally begun as an immense hotel, and now constituting the largest educational edifice of its class west of New York), and the adjoining magnificent stadium, constructed in concrete, and capable of accommodating 25,000 persons.

At the Tacoma smelter, the party was received by Messrs. F. W. Clark, manager, and Roger Taylor, superintendent.

The Tacoma smelter treats both lead- and copper-ores—the aggregate of both for the year 1908 having been 189,898 tons, of which 115,666 tons came by water, and 74,232 by rail—an exhibit which sufficiently indicates the advantage of this site in commanding supplies from a thousand miles of coast. Ores were received from Washington, Oregon, California, Nevada, Idaho, Montana, Alaska, British Columbia, Mexico, South America, Korea, Japan, and the Philippines; and copper-matte from other smelters in the Northwest not equipped with converters for treating that product.

There are two sampling-mills, for the lead- and copper-departments respectively, and an extensive assay-office and laboratory.

The lead-smelting department contains three large mechanical roasters (capacity, 45 tons each per day); a high stack, a briquetting-plant (150 tons per day), and four blast-furnaces, of whicb the two now running smelt daily 300 tons of charge,

and produce 40 tons of lead-bullion. The furnace-fumes are conducted through a flue into a large chamber containing 1,000 cotton bags, through which they are filtered, to recover the solid particles of flue-dust, for sintering and subsequent smelting.

The copper-smelting department contains two water-jacketed 14- by 3.5-ft. blast-furnaces, each smelting 375 tons per day; and two converters in constant operation, with a daily capacity of 30 tons of matte each. The blister-copper from the converters, containing about 99 per cent. of copper, is delivered by ladle to a casting-machine and cast into 350-lb. ingots, for further treatment in the refinery, in which the precious metals and the impurities are removed.

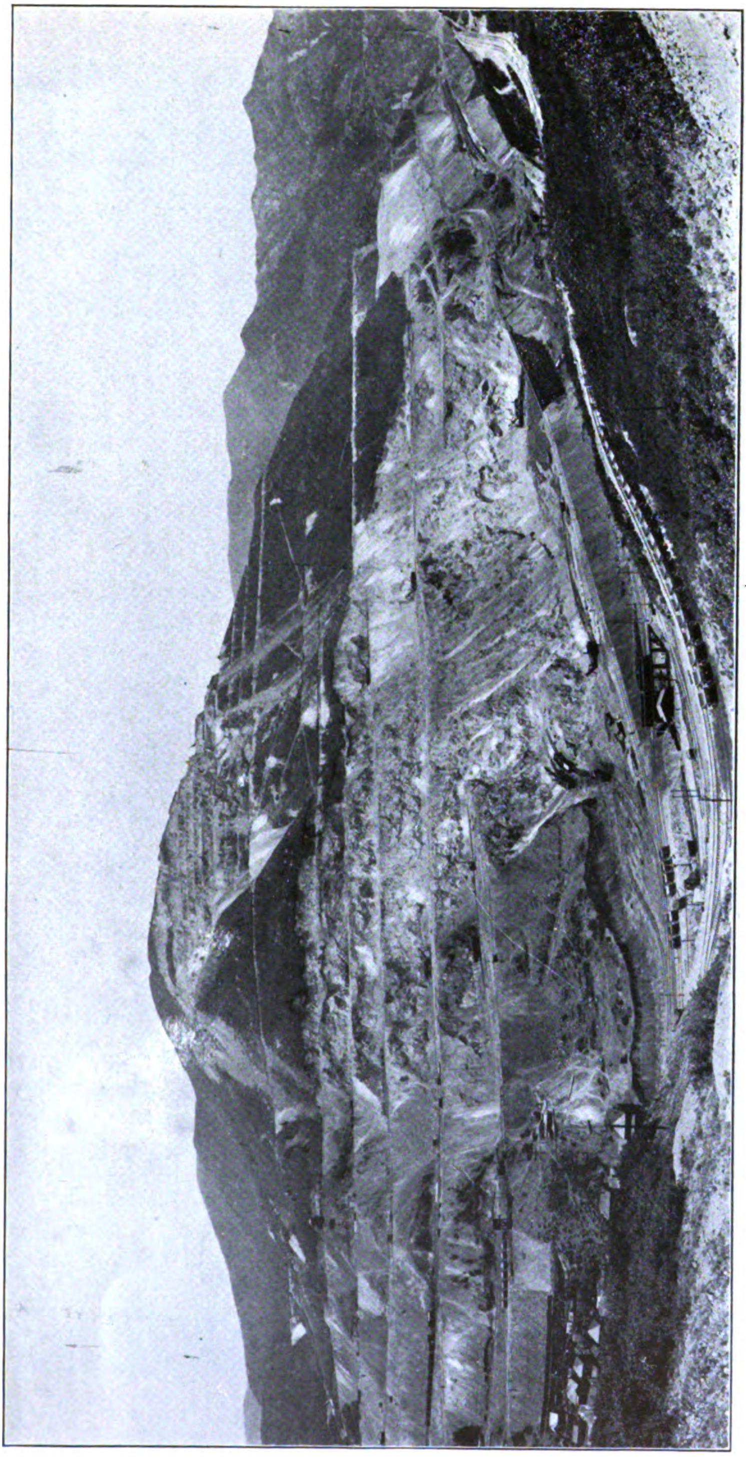
Meanwhile, a portion of the party, including the ladies, were conveyed to the famous Point Defiance Park, a wilderness of primeval forest, including zoological and botanical gardens, and many picturesque features. Here they were rejoined by the rest, and the whole company, embarking in a steamer, enjoyed a memorable evening sail along the extensive water-front of the city, with sunset light on Mt. Rai—no, Tacoma! —in the distance. During the night, the train was run to Portland.

Portland and The Columbia.

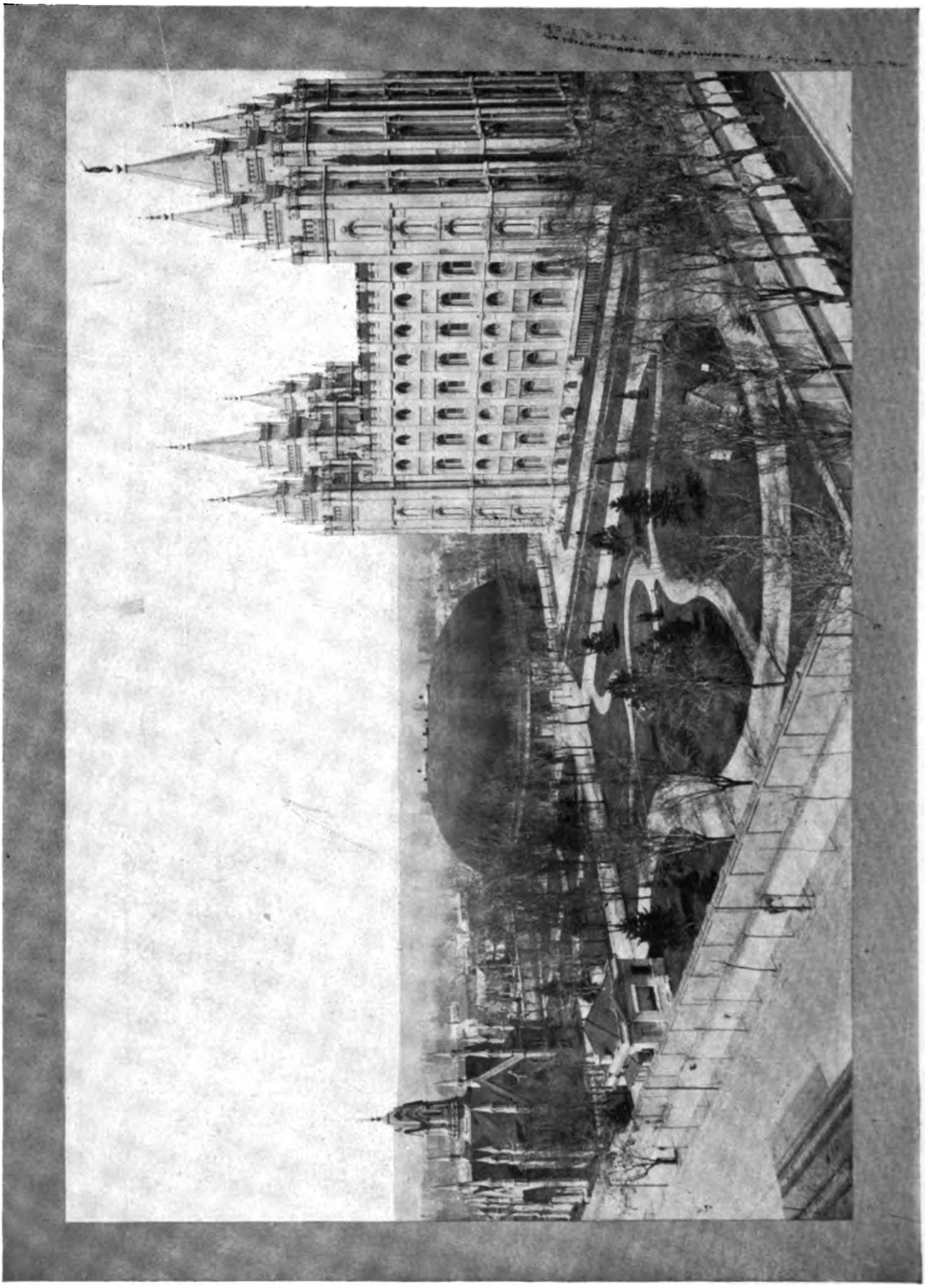
Tuesday, Oct. 5, was the first and only really disagreeable day of the journey. It was dismally dark and rainy; and the great majority of the party declined to go on the already crowded steamer from Portland to the Dalles, preferring to stay in the former city for a few hours and then to proceed to the Dalles upon their special train, over the railway along the bank of the river. A few, who braved the discomfort of the trip, found it not intolerable, and were rewarded by their view of this most majestic of rivers, bordered by autumn colors—not gay like those of the Hudson, but fierce and startling, like the yellow and brown and black on a tiger. When the two divisions were reunited at the Dalles, just after dark, each envied and pitied the other; and the question, whether they who viewed the whole Columbia from a damp deck, or they who looked upon it from one side, in a dry and comfortable train, were most to be congratulated, is still open to discussion. As is well known, the Institute favors full discussion, but avoids authoritative decision.

Salt Lake City and Vicinity.

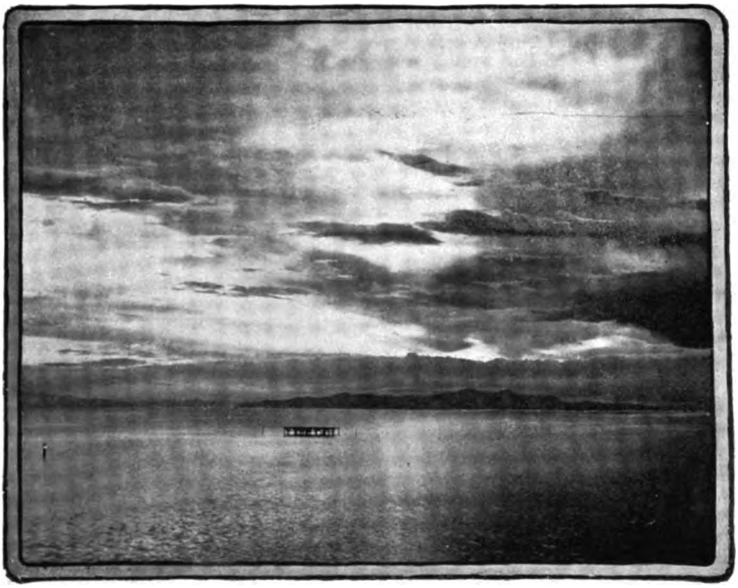
Wednesday, Oct. 6, was rainy, but not disagreeable on that account; for the day was spent in transit over the Oregon



UTAH COPPER MINE, BINGHAM, UTAH.



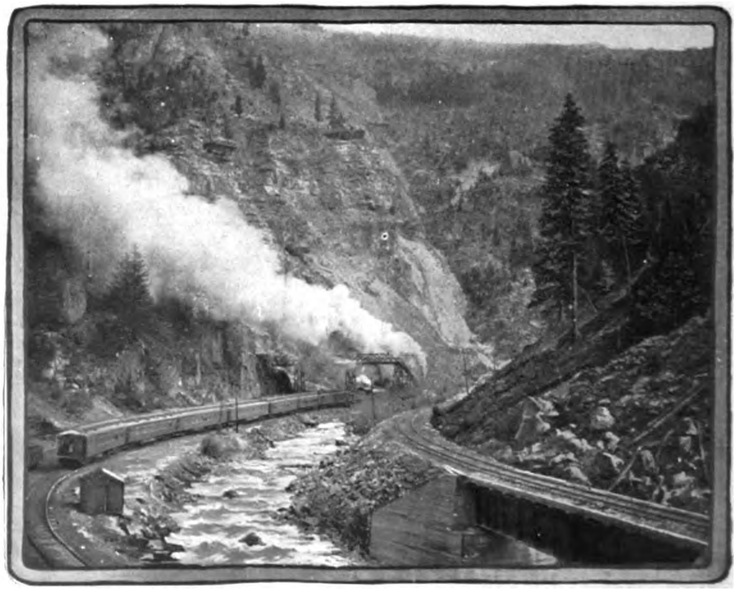
TEMPLE AND TABERNACLE, SALT LAKE CITY.



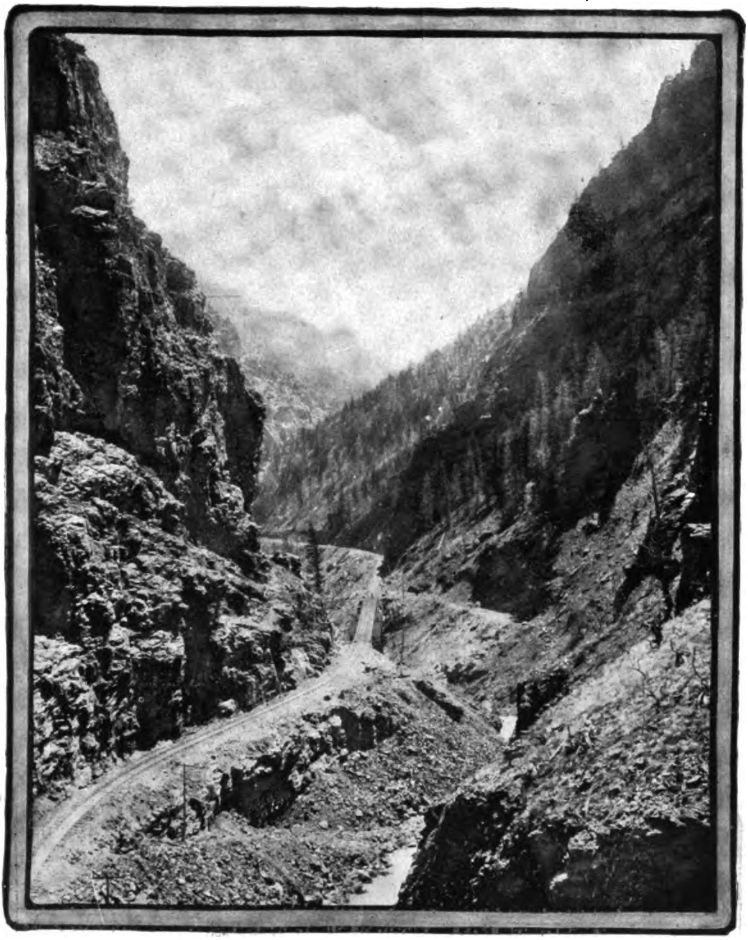
SUNSET, SALT LAKE.

CASTLE GATE, UTAH.

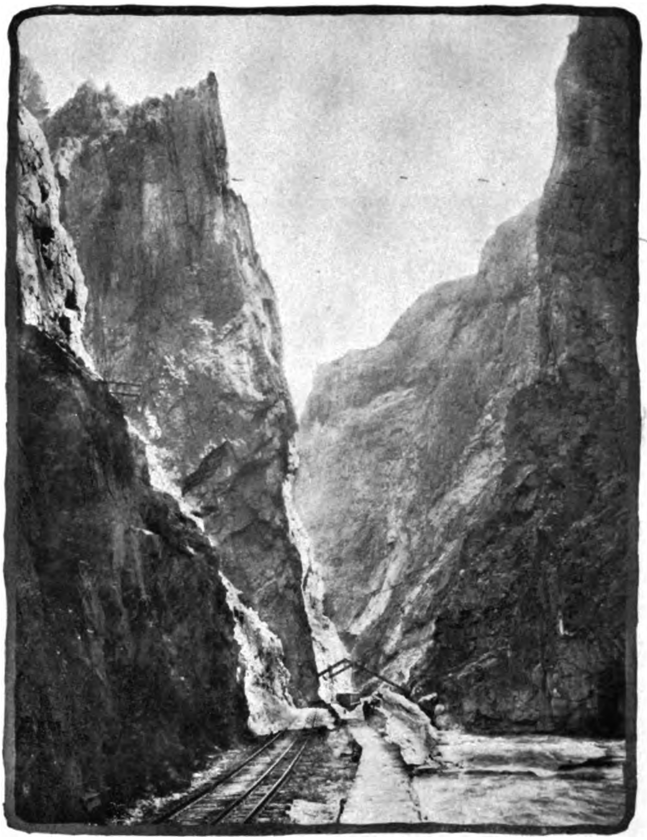
A TYPICAL COLORADO LAKE.



EAGLE RIVER CAÑON, COLO.



BLACK CAÑON OF THE GUNNISON.



ROYAL GORGE, GRAND CAÑON OF THE ARKANSAS.

GATEWAY, GARDEN OF THE GODS.

Short Line from the Dalles across the lava-country, along the cañon of the Snake, and over the divide (where once Lake Bonneville emptied its waters into the Columbia drainage-system) to the Salt Lake valley. All the way were visible the evidences of the successful cultivation of what was formerly regarded as worthless desert. And the rain laid the dust for us, completing the unusual record of our long journey, during which, contrary to precedent and anticipation, we had not once been seriously incommoded by dust. We reached Salt Lake City late at night.

On Thursday, Oct. 7, we were again blessed with fair weather, and our train was taken over the Denver and Rio Grande road to Bingham City. At that point a Shay engine with open platform-cars was at hand to convey the party up Bingham cañon over the Copper Belt railroad and Bingham high line. This trip, which carried us almost up to the snowy tops of the Oquirrh range, was extremely picturesque (and somewhat perilous, the grade of the road being in places 8 per cent.), and afforded opportunity to see many novel features in mining, including the aërial tramways, which transported, high in the air, their freight of ore (and now and then even a man), and, above all, the amazing caving and steam-shovel mining-operations of the Boston Consolidated Mining and Utah Copper Companies.

The copper-bearing deposits worked by the Utah Copper Co. and the Boston Consolidated Mining Co. in Bingham cañon are due to the mineralization of certain zones in a body of intrusive monzonite porphyry, inclosed in quartzite. The Utah Copper Co. began operations in 1903, adopting at first a caving system. In 1906, the Boston Co. started to mine its ore-body with steam-shovels. At present, the Utah is mined chiefly by steam-shovel and the Boston by caving.

The Utah Copper Co.

Of the area of about 200 acres embraced in the patented mining-claims held by the Utah Co., about 80 acres is known to be ore-bearing. According to the report of the general manager, dated February 1, 1909, the property presented at that time 20,000,000 tons of fully developed, and 60,000,000 of partially developed, or reasonably assured, ore. (These estimates include, however, what had been already mined—which was about 3 per cent. of the total of 80,000,000 tons). About 65,000,000 tons are estimated as averaging 2 per cent. and the remaining 15,000,000 as carrying 1.5 per cent. of copper. There is also a lower zone, the extent and average value of which have been indicated to a limited extent by diamond-drilling, on the strength of which it is estimated to contain at least 40,000,000 tons of ore, which will probably yield 1.5 per cent. of copper. The

mine-equipment comprises 10 steam-shovels, 21 locomotives, several hundred cars, and 16 miles of track.

During 1908, 1,881,818 tons of ore were mined, containing 13,366 oz. of gold, 124,640 oz. of silver, and 41,924,834 lb. of copper, the gross yield being valued at $5,850,765. In 1909, the product was larger. At the time of our visit, the mine was said to be shipping 10,000 tons of ore daily; and the official report for the third quarter of 1909 shows an average increase in the daily product of ore of about 100 tons over the figures for the preceding quarter. It is impossible, however, to make a just comparison without a careful analysis of details, such as cannot be attempted here; for the work of stripping, preparatory to steam-shovel mining, varies from month to month.

The Utah Co. has two reduction-plants: one at Copperton (capacity about 900 tons per day), situated below Bingham in the cañon; the other at Garfield, about 15 miles north of the mine, and near the shore of Salt Lake. The Copperton plant was not visited by the Institute party. Although it is confessedly less economical than that at Garfield, it is still operated by the company, and improvements are in progress which are expected to bring it more nearly into line with the newer plant.

The Garfield plant of the Utah Co. is constructed of steel and concrete, in 12 sections, each complete in itself and capable of treating independently 500 tons of crude ore daily. The main building is 600 by 508 ft. in size. The plant is, in a general way, divided into three departments: coarse crushing, fine crushing, and concentration. The coarse-crushing department comprises two divisions, each with a capacity of 3,000 tons in 16 hours, and each equipped with two No. 7½ gyratory crushers, and two belted rolls 54 in. in diameter by 20 in. face, by which the ore is crushed dry to 0.75-in. size. The fine-crushing department contains thirty-six 6-ft. Chile mills and 24 belted rolls 36 in. in diameter and 16 in. face. Here the ore is crushed wet to 40-mesh size. The concentrating department contains 48 Wifley tables, 1,104 six-foot suspended vanners, 24 "Garfield roughing-tables," 24 four-compartment classifiers, and 384 conical settling-tanks, together with the necessary elevators, etc. All the concentrating-machinery is set on reinforced-concrete floors, supported by steel columns, and all the launders for tailings and concentrates are below the floors. The total area of concrete flooring slightly exceeds 5.5 acres, and the total area covered by the main mill building and the ore-bins is about 7 acres. The receiving-bin into which crude ore is delivered from bottom-dump railroad cars holds 25,000 tons, and the receiving-bin for coarse-crushed ore holds about 15,000 tons, making a storage-capacity of 40,000 tons, or about 7 days' supply for the plant.

The power-plant comprises 20 water-tube 600-h.p. boilers, set in pairs; 5 cross-compound condensing engines (2 of 1,200 kw. and 3 of 2,000 kw. capacity), all directly connected with 100-revolution, 4,000 volt, alternating-current generators. Current is transmitted to the mill at the above potential, and there transformed to 440 volts for use in the plant. The transmission-line to Bingham, of stranded copper cable, supported on steel towers, 400 ft. apart, carries 40,000 volts, which is transformed to 440 volts for use at the mine and at the Copperton mill.

The Boston Consolidated Mining Co.

The Boston Consolidated Mining Co. owns 378.8 acres, containing two separate mines, the Porphyry and the Sulphide. The Porphyry has 101,914 feet of underground workings, and is estimated to contain 37,000,000 tons of porphyry ore, carrying more than 1.5 per cent. of copper. It is now worked by caving,

but is equipped for steam-shoveling. It employs 427 men, and produced in September of this year 71,880 (dry) tons of ore at a total mining-cost of 66.7 cents per ton, as follows: Breaking ore, 18.8; mucking, 16.5; timbering, 7.7; electric haulage, 4; gravity train, 2.5; mine-expense, 5; general expense, 2.2; develment, 10 cents. The Sulphide mine has 47,536 ft. of underground workings, uses the square set stoping method, employs 102 men, and produced in September 7,178 dry tons of ore (2.16 per cent. copper) at a total mining-cost of $1.688 per ton, divided in cents as follows: Breaking ore, 29.1; mucking, 35.2; timbering, 50.1; electric haulage, 9.6; general expense, 4.1; mine expense, 12; development in barren material, 28.7. Adding the men in the shops, the aggregate number for both mines is 548. The equipment for both includes a 4-track gravity tram, 2,000 ft. long, vertical height 737 ft., 24 to 55 per cent., with 12-ton cars—daily capacity, 6,000 tons; a 3,000-ton steel receiving-bin; four 90-ton steam-shovels; 12 locomotives; 146 railroad cars; 6 miles of narrow-gauge track; 3 electric-driven air-compressors, of 70 drills' capacity; 80 rock-drills; shops, warehouses, boarding-houses, etc.

The mill-site of the Boston Consolidated Co. has an area of 910 acres, and the plant comprises the following:

Foundry.—Steel building, 60 by 80 ft, with motor-driven Whiting cupola, brass-furnace, Roots blower, and 12,000-lb. jib crane.

Machine Shop.—Frame building, 44 by 178 ft., with motor-driven 10-ton traveling crane, lathes, cutting-, boring-, drilling- and shaping-machines, 1,100-lb. steam-hammer, etc.

Electric Sub-station, Transformer Building.—Concrete and brick, 60 by 72 ft., with the following Westinghouse transformers: Four 1,000-kw., 80,000–460 volts; three 250-kw., 460–2,300 volts; three 75-kw., 460–115 volts (for lighting); also, 80,000 volt choke-coil and arrester, and a 9-panel distributing switchboard. Power is transmitted at 40,000 volts, 144 miles, from the Telluride Power Co.'s station at Grace, Idaho.

Crude-Ore Bin.—Steel, 36 by 300 ft., capacity 12,000 tons.

Crusher-House.—Steel, 66 by 78 ft., with two No. 6 and two No. 5 Gates gyratory crushers, and screens, conveyors, etc., all motor-driven.

Mill.—Steel, 326 by 590 ft., with crushed-ore bin, 21 by 570 ft., and motor-driven equipment as follows: 312 Nissen stamps (1,500 lbs.; 6.5 in. drop; 114 drops per min.; capacity, 8.73 tons per stamp-day); 299 Wilfley tables, 221 Johnston vanners, 312 Callow tanks, classifiers, launders, etc. The pulp flows from the stamp through the mill by gravity, and tailings and concentrates are conveyed away from the mill through a tunnel having 13 branches underneath the building.

Heating-Plant.—Frame and iron, with three 80-h.p. boilers.

Main Pumping-Plant.—Brick, 30 by 50 ft., with one 3-stage Byron Jackson centrifugal pump (3,000 gal. per min.), and two 4-stage Wood centrifugal pumps (each 1,500 gal. per min.), all working against static head of 380 ft. through 4,316 ft. of 20-in. wood pipe-line.

Concentrate-Bins.—Nine 200-ton concrete bins, from which concentrates are loaded into cars by Browning crane.

Assay-Office, Store-Room, etc.

Number of Workmen.—Foundry, 15; shops, 21; electrical department, 6; unloading ore, 10; crusher-house, 18; mill, 114; loading concentrates, 3; clerical force, 5; assay-office, 2; outside labor, 34; total, 228.

Results in September, 1909.—Received 77,003 tons of ore; milled 72,000 tons (dry); produced 3,550 tons (dry) of concentrates, containing 1,569,000 lb. of copper, recovering 21.8 lb. copper per ton of crude ore. Average of heads, 1.58 per

cent. Power required: For pumping, 469,280; for milling, 989,265; total, 1,458,545 kw-hours. Possible stamp-hours, 224,630; actual, 214,145, or 95.3 per cent.

Milling-Costs.—The costs for the month of September were, in cents per ton: Crude ore-bin, 1.09; crushers, 4.49; stamps, 21.97; Wilfley tables, 2.45; vanners, 3.35; launders, 0.51; Callow tanks, 1.01; loading concentrates, 0.44; tailings-dump, 0.74; water-supply, 7.50; heating-plant, 0.04; mill expense, 3.41; general expense, 3.11; total, 50.11 cents.

On the return trip from Bingham stops were made at Bingham Junction, where the United States smelter was visited, and at Murray, where a hurried inspection of the smelter of the A. S. & R. Co. was made.

The United States smelter, situated about 12 miles south of Salt Lake City, is treating, in 6 water-jacketed 45- by 160-in. blast-furnaces, from 800 to 900 tons daily of lead-ores from the Jordan and the Galena mines in Bingham cañon, from Tintic district and from Eureka, Nev., etc. The two most important and novel features of the works are: (1) The Huff electrostatic method of separating the iron-zinc concentrates of the ordinary wet concentrator into an iron product, carrying the precious metals, and a high-grade zinc-blende; and (2) the method employed for treating the furnace-gases and fumes. This is done by first "neutralizing" with volatilized zinc-oxide and certain cheaper chemicals such gases as carry sulphur trioxide, corrosive sulphates, or free sulphuric acid, and, after cooling all the gases, blowing them through a bag-house, in which they are strained through bags (cotton for the blast-furnace gases, and woolen for the roaster-gases, which are more corrosive). The bag-house product, about 20 tons daily, contains mainly lead, silver, and arsenic. The arsenic is recovered and refined in a separate plant, and the lead and silver are extracted from the sinter. After being thus freed from deleterious solids the gases are allowed to escape through chimneys of moderate height, after diluting the sulphur dioxide which they still contain, to less than 0.75 per cent. of their total volume. It is claimed that this is not injurious to vegetation, and this claim has been proved to the satisfaction of the U. S. Court, which in August, 1909, upon evidence of the success of the method with lead-ores, removed its injunction against the treatment at these works of copper-ores also.

In the evening, through the courtesy of President Joseph H. Smith, of the Church of the Latter-Day Saints, the party was entertained at a private exhibition of the new organ in the Tabernacle. This wonderful auditorium has been renovated within a few years past, and now presents a very handsome interior. The famous Tabernacle organ, originally built in Salt Lake City, and always noted for its sweetness and power, was reconstructed nine years ago by the W. W. Kimball Co., of Chicago. Some of the original pipes still remain, but the instrument is new in the main, and is said to be the largest save one in the United States. Certainly, the Tabernacle organist,

Prof. John McClellan, aided perhaps by the acoustic properties of the great auditorium, produced a range of musical effects from it which surpassed anything within the experience of his listeners. It was a memorable, entrancing hour.

On Friday, Oct. 8, a trip was made by the San Pedro Railroad to the different smelters and concentrators in the vicinity of Garfield, including the Garfield smelter of the A. S. & R. Co., and the concentrating works of the Utah Copper Co., and the Boston Consolidated Copper Co., already noted.

Before returning to Salt Lake City, a visit was made to the International mill and smelter under construction at Tooele.

On Saturday, Oct. 9, under the guidance of the Utah Society of Engineers, a trolley-trip, extending as far as Fort Douglas, was taken, and in the afternoon the pavilions and bathing-houses of Saltair Beach were visited. The season was over, and the place seemed deserted; but many of the visitors seized the opportunity of bathing in the dense waters of Salt Lake—much to their own amusement and that of those who looked on. On the way home, a glorious sunset, illuminating the snowy peaks and precipices of the Wasatch range, furnished an impressive farewell picture of the beautiful Salt Lake valley.

Glenwood Springs.

Leaving Salt Lake City at 8 p.m. by the Rio Grande and Western route, and passing in the night the interesting and typical scenery of Castle Gate (but something must be omitted, if folks are to sleep at all!), we reached at about noon of Sunday, Oct. 10, the famous and beautiful watering-place of Glenwood Springs, Colorado. The fashionable season was over, and the great hotel was closed; but the swans were still swimming on the little pond in the courtyard, and the famous Blue Pool and steaming caves were still accessible to bathers, as the party proved to its merry satisfaction. After two hours' stay in this fascinating locality, the train proceeded eastward through the magnificent cañons of the Grand and Eagle rivers, past the precipices of Red Cliff (crowned with the relics of mining-enterprises), and finally through the Hegeman tunnel, over the Tennesee pass, at an altitude of 10,200 ft. (the highest railway-tunnel in America), into the great Arkansas valley, bordered by the magnificent snowy Sangre de Cristo range, and across this

valley to Leadville, which was reached at 8 p.m., after a day favored with perfect weather and filled with the enjoyment of sublime scenery. Notwithstanding the lateness of the hour, a considerable portion of the party accepted the cordial invitation of our Leadville members, and visited, under the guidance of John R. Champion, the great Yak tunnel, more than two miles long, which serves several important mines with drainage and transportation.

The Royal Gorge.

At 3 a.m. on Monday, Oct. 11, the train left Leadville, reaching Salida at about 6, and thus permitting the party to see by daylight the lofty snow-covered peaks of the Sangre de Cristo range before threading, through the deep cañon of the Arkansas, the "Front range" of the Rockies. This passage was, perhaps, the most impressive part of the whole of our tour. The transit through the Royal Gorge is made by tourists, in most instances, from east to west. At least that is the experience of the writer, who has often made it. But the journey in the opposite direction, especially when it takes place (as in this instance) in the early morning, is much more dramatically impressive, because the climax comes at the end. In traveling from Cañon City westward the deepest and sublimest point in the Gorge is soon reached, and, after that, the scenery dwindles in picturesque significance until, upon emergence into the broad Arkansas valley below Salida, it is replaced by the totally different panoramic view of the Sangre de Cristo range. But the eastward traveler, after losing sight of that majestic range as a whole, is refreshed by occasional glimpses of its snowy summits while he plunges into the western foot-hills of the Front range, and finally, after the last glimpse of those shining peaks, is seized by the fascination of the ever-deepening and more and more precipitous cañon in which the Arkansas breaks through the Front range, to pursue its thereafter peaceful course. In early morning, when the sun has risen upon some portions of the cañon, while others are still shrouded in shadow, the progressive sublimity of the scenery is doubly grand.

The Royal Gorge is the most sublime piece of scenery in Colorado—unless the palm be given to the Black Cañon of the Gunnison, which, being traversed by the old Marshall Pass

narrow-gauge line of the D. & R. G., is not accessible to a broad-gauge special train. It is reported that this part of the track will soon be changed in gauge. Meanwhile, no one can say he fully knows the scenery of Colorado, who has not traveled on the D. & R. G. narrow-gauge system. Another feature, which railroad-travelers miss, unless they seek it by side-trips, is furnished by the mountain-lakes, of which a typical picture is given herewith.

Pueblo and Colorado City.

At 11 a.m., on Monday, Oct. 11, the party arrived at Pueblo, Colo., where it was divided—a part, including the ladies, going on with the train to Colorado Springs (from which point Manitou, the Garden of the Gods, etc., were visited), and the remainder, under the efficient guidance of the Pueblo Local Committee, visiting the Minnequa plant of the Colorado Fuel & Iron Co., the works of the United States Zinc Co., and the Pueblo lead-smelting plant of the A. S. & R. Co. The Denver & Rio Grande R. R. Co. courteously furnished for the latter purpose a special car and engine; and, lunch being served on the car in transit, no time was lost in carrying out the program of the day.

Homeward Bound.

At 7 p.m., Oct. 11, the party, reunited upon its own special train, left Colorado Springs for Kansas City and Chicago. This prairie trip was enlivened socially by many interludes, including a "mock trial" of most hilarious nonsensical character. Such things cannot and will not be officially recorded, partly because their delightful atmosphere would be lost in any attempt to depict them, but chiefly because, according to the immemorial tradition of the Institute, members who are so foolish or so unfortunate as to miss them are not entitled to learn, at second-hand, anything about them!

On Wednesday, Oct. 13, the party reached Chicago, and one of the most thoroughly delightful and instructive excursions of the many which have illuminated the history of the Institute ended in mutual congratulations and parting regrets.

The writer may be permitted, in conclusion, to offer some observations, neither scenic nor social, concerning this long

and happy journey. Probably no member of the party could have anticipated less in the way of novelty from such a trip than one who had already crossed the Continental Divide, back and forth, more than fifty times, who had conducted some of the earliest explorations in the West, and who had been for more than forty years associated with the development of its resources. In fact, he entered upon this journey somewhat *blasé*, with the feeling that he was to traverse a familiar track, and must find profit and enjoyment chiefly in sympathy with less experienced traveling-companions, to whom everything would be new. But this expectation was destined to be lost in a startling surprise. Of his innumerable former professional journeys, not one ever brought before him so many features of novelty as well as importance. Many of them had become partly known to him through report; but seeing is more than hearing; and the following list of those things which most deeply impressed a "veteran observer" may interest and encourage his younger colleagues, who are actively engaged in the work from which he has practically retired:

1. The enormous recent industrial development of the West, from the Rocky mountains to the Pacific, is an astounding marvel, especially to one who had dreamed of it, hoped for it, and seen it coming, but did not conceive that it could come so soon and in so many places at once.

2. The stimulation of this development by the irrigation of what was once regarded as irreclaimable desert-land, and by the utilization of water-powers through electric transmission, is another source of surprise. For it must be remembered that, in these departments, American genius has not merely copied the methods of other countries, but has also improved them, finding its own solution for its own problems, and, in many instances, setting an example of daring and ingenuity for the world to follow.

3. The consolidation of capital in large enterprises has permitted the employment of scientific ability and the introduction of improvements beyond the reach of individual enterprise, which have established permanent industries, giving employment to thousands of workmen, and assuring the prosperity of large communities. These aggregations of capital, commonly but incorrectly called "trusts," have been the object of much

thoughtless and indiscriminate denunciation. No doubt they should be, like all other associations of citizens, subject to the law; but whether they should be condemned *in toto* is a different question; and on this question, evidence is admissible. Such evidence is furnished by the following, among many other, facts of industrial progress:

a. The economies in mining-operations and in mining-administration effected in Butte through the consolidation of mine-ownership.

b. The establishment of the great Anaconda smelting-works, treating daily 10,000 of the 15,000 tons of ore daily produced by the mines of Butte. Little, if any, of this enormous product could be profitably mined at all, but for the provision thus made, by a so-called "trust," for its cheap concentration and economical treatment.

c. The remarkable improvements, made by Mr. Mathewson and others, in the handling and treatment of low-grade copper-ores, including the matte-smelting in very long reverberatories, and the more recent enlargement of the horizontal length of blast-furnaces. As to each of these improvements, pages of discussion might be offered. But it is sufficient to say here that they have been recognized and copied as good; that they have effected the economical treatment of ores otherwise worthless; that they have consequently continued the employment of thousands of workmen; and that these results would not have been realized, but for the existence of so-called "trusts," willing to employ scientific aid and able to pay for it.

d. The wonderful development of mining-operations in Bingham cañon, Utah, as shown especially in the operations of the Utah Copper and the Boston Consolidated Companies. Concerning these enterprises, some details have been given above. They present what is (with the possible exception of the Rio Tinto in Spain) the largest productive mines in the world. The simple statement that the Utah Co. ships 10,000 tons of ore a day, and would have made, a week before our visit, if the railroad had not broken down in its attempt to furnish the necessary cars, a "record" shipment of 20,000 tons in one day, together with the further statement that a single blast in the face of its mineralized zone loosened and dislodged for the work of the steam-shovel 66,000 tons of material, will serve to

show the revolutionary character of this new style of mining. But when it is added that the material thus mined contains 2 per cent. or less of copper per ton, a new question is suggested. The writer remembers visiting in Bingham cañon, many years ago, a member of the Institute, who was engaged in mining and smelting certain lead-bearing surface-ores of that district; whose residence stood upon the outcrop of this low-grade copper-bearing belt; who was perfectly aware of the presence of 2 per cent. of copper in the rock under his doorstep; but who was neither foolish enough to persuade himself nor dishonest enough to persuade other people, that a 2 per cent. copper-ore could be at that time mined with profit. What has produced the change in these conditions? There can be but one answer: It is the employment of capital in the hands of men willing to pay for scientific aid, and to take risks on scientific advice, and able to make large expenditures in such adventures.

e. The correspondingly immense establishments for the concentration and subsequent treatment of low-grade ores, of which some notice has been made above, including the Anaconda works, which receive 10,000 tons a day, and the various large establishments in the Salt Lake valley, handling the output of the Utah, Boston, and other mines. Under the head of size alone, these establishments are overwhelmingly impressive; but to one who knows what difficulties of design and operation are involved, these triumphs of skill and judgment seem doubly marvelous. Here again it is the representative of capital, willing and able to follow expert guidance and to take commercial chances, who has "saved the situation," by providing, at great outlay and risk, the means and methods for the treatment of low-grade ore in large quantities.

f. In the same category belong the many recent improvements, metallurgical or mechanical, which characterize present American practice in the West—such as the electro-static separation of zinc-blende and chalcopyrite by the Huff process at Bingham, Utah; the treatment of a similar problem at Pueblo, Cal., by a partial roasting and a subsequent magnetic separation; several ingenious features in the feeding and management of the open-hearth furnaces of the Colorado Fuel & Iron Co., Pueblo, Colo.; and, last but not least, the remarkable electric mine-hoist of the Hecla mine, in Idaho. All these and

many other technical improvements which may not have come to the notice of the writer, or may escape his present recollection, are directly due to the intelligent employment of large amounts of capital.

4. The writer would, therefore, sum up the profound impressions derived from this memorable trip by saying that, to his mind, they furnish an astonishing exhibit of the work of private enterprise in the West, during the last few years, and constitute a strong argument in favor of that system under which the West has made such unprecedented progress.

[SECRETARY'S NOTE.—In the preparation of the foregoing sketch, I have been indebted for many particulars which might have escaped my notice at the time, or my recollection afterwards, to the diary of Mrs. W. S. Mitchell, kindly placed at my service. In one respect, this faithful record, daily made under circumstances which would have discouraged a "mere man," has been invaluable—namely, it has furnished me with unimpeachable evidence as to the weather, and thus with the means of confounding those pessimistic members who did not go with us to the Yellowstone Park and Pacific Coast, etc., because "the season was unpropitious."

Special acknowledgment is due also to the Northern Pacific R. R. and the Denver and Rio Grande R. R. for many of the engravings with which the foregoing narrative is illustrated, and to the officials of various mining and smelting companies who furnished the material from which the interpolated brief sketches of mines and works have been prepared. Many omissions in this department have been due to the non-arrival of such information before it was necessary to put this report to press.—R. W. R.]

In accordance with the established practice of the Institute (replacing the unsatisfactory method of passing an "omnibus" resolution of thanks), separate official letters have been sent by the Secretary to all persons and organizations concerned in the entertainment of the Institute party. In addition to these official recognitions, the members of the Institute traveling-party have presented to each of the following gentleman—Charles W. Goodale of the Butte Committee, E. P. Mathewson of the Anaconda Committee, L. K. Armstrong of the Spokane Committee, Chester F. Lee of the Seattle Committee, and Duncan MacVichie of the Salt Lake Committee—a gold watch-fob (resembling those illustrated on p. 966 of the *Bulletin* for November, 1906), in acknowledgment of their special labors for its reception and entertainment. A similar watch-fob has been presented to Mr. Theodore Dwight, who, although unable to

accompany the party, conducted the initial arrangements of organization. And a "flat" silver table-service, comprising 36 pieces of a beautiful "Florentine" pattern, was presented to Mr. Albert E. Vaughan, who so vigilantly and skillfully conducted the excursion from beginning to end as to make that success complete, in spite of numerous unforeseen obstacles and necessary changes of the details of the pre-arranged schedule.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Piping and Segregation in Steel Ingots.

Discussion of the paper of Mr. Howe, *Trans.*, xxxviii., pp. 3 to 108; 924 to 935; xxxix., 818 to 850.

P. H. Dudley, New York, N. Y. (communication to the Secretary*):—This renewed discussion of Professor Howe's paper is partly due to a recent conversation with him, in which I called attention to some features of modern practice he had not stated, and he asked me to state them for publication, that eventually the truth might be ascertained. His invitation to present observations and their interpretations, as applied in practice, is accepted in the broad spirit in which it was extended.

My observation during many years of practice in teeming steel ingots has been that piping or shrinkage-cavities and segregation are greater in the higher-carbon steels than in the medium and mild steels. Hence we are obliged to discard a larger portion of the ingot of sound high-carbon metal, especially as its dimensions are increased. The necessity of teeming all kinds and grades of steel involves the question of the greatest percentage of sound or available metal free from pipes and sponginess, whether of crucible, Bessemer, open-hearth, or electric manufacture, which can be used for the purposes intended. This question requires renewed investigation, in order that we may secure a better and higher grade of steel, as the metal is subjected to more severe service in the rapid progress of the industrial arts.

Many members of the Institute can recall the rapid failure of the iron rails from 1860 to 1865, when the driving-wheel loads of the engines reached from 10,000 to 12,000 lb. The physical properties of wrought-iron, with its 1 to 1.5 per cent. of included cinder, had been sufficient for the evolution of the railroads, but proved inadequate for their subsequent development. The substitution in 1865 of light Bessemer steel sections for iron rails was facilitated by the fact that the steel, having been

* Received Sept. 10, 1909.

once molten in manufacture, would be more homogeneous than iron.

All of the early Bessemer rails were rolled from ingots which, after being teemed and "stripped in the pits," were allowed to cool for examination, as in crucible practice, to see whether they were suitable for rails. The ingots selected were reheated and hammered into blooms; and many cases of genuine piped rails with smooth unwelded walls of the web, from cold-shrinkage cavities, were subsequently found in the tracks.

In the winter of 1876–77, I was sent to investigate a number of piped rails which had split in the web and broken in the track. In a few cases I found slag between the walls, but in most cases the walls were oxidized. I went also to the mill in which the rails were made and found ingots which had been allowed to cool, of which 80 per cent. were piped at one or both ends. This fact, in connection with some forcible language by the general manager of the mill, in answer to my question, "whether this was an extraordinary percentage, or the usual amount of piped blooms," made a lasting impression on my mind. I concluded that the cavities represented the interior volume of shrinkage from the molten metal of teeming to the cold metal of the ingot-walls, and that the greater part of it could be avoided in rail-ingots by charging them into the reheating-furnace as soon as possible after they were stripped. This method soon became the regular practice at all rail-mills, and the percentage of piped rails from shrinkage-cavities was reduced. Professor Howe does not give this feature of the "state of the art."

I have always worked upon the plan of checking the full amount of the interior shrinkage of the volume of the steel in the ingots for rails by prompt reheating after stripping, in connection with the chemical composition required for the physical properties and degree of deoxidation desired for the grade of steel made.

The adjustment of the chemical composition, first, to secure the proper physical properties, and, secondly, to obtain as sound ingots as practicable under the existing conditions of manufacture, was coincident with my design and use of heavy sections. I did not expect good results in manufacture unless I made proper provision to secure them; and the freedom from piped

rails in more than 500,000 tons, which I made from 1891 to 1897 inclusive, is evidence to me that the theory upon which I worked had basis of fact. I can now control my chemical composition for the section and size of ingot, and with proper time for the teeming, stripping, and charging into the reheating-furnaces, can secure practically pipeless rails. My statements refer to rail-ingots, and to secure the desired results my specifications are not universal, but have always had an adaptability to meet local conditions of manufacture at different mills.

In teeming ingots for basic open-hearth rails, the steel does not, as a rule, set "dead," as it does in Bessemer practice, but the escaping gases eject sparks and the metal rises in the molds. A cast-iron plate is often placed on the top of the steel, and in some mills the top of the mold is capped and keyed. Aluminum is often used on top of the metal in the molds to quiet basic open-hearth steel for plates as well as for rails. Bessemer steel of from 0.10 to 0.30 per cent. of carbon for billets, splice-bars, and bolts is often teemed in bottle-mouthed molds which are not completely filled, but are capped and keyed to prevent the rising metal from overflowing the molds before it freezes. In teeming the ingots of the various kinds and grades of steel there are distinctive methods of practice, which have been evolved after years of experience in producing the several grades of steel required in service.

Professor Howe, studying the shrinkage-cavities or pipes from cold ingots in which the entire shrinkage of the metal from the teeming-temperature to that of cold steel has occurred, of course finds conditions of manufacture which require attention and correction. Operating-men at the mills and engineers have also studied the obstacles to be overcome, and applied remedies with such success that the conditions are decidedly better than they would seem to be from a study of the cold ingots. There is still much to be done in the way of improvement and progress, which will continue as long as steel is made and teemed into ingots. A method has not yet been discovered by which an ingot can be teemed, its non-piping period checked or prolonged at one stage of the process, and then cooled so as to retain what had been gained, as though the entire manufacture, as far as the metal was concerned, had been completed before the steel cooled. A good estimate of the effectiveness of a

method of checking the shrinkage-volume can be made thus: Take an ingot from a heat, allow it to cool, and then cut and measure its cavity; then take a bloom-crop, the discard from an ingot of the same heat, and cut it. In good practice the shrinkage-cavity will be but a small percentage of that found in the cold ingot, and will indicate the reduction secured.

Fig. 1 is a photograph of a three-rail ingot, for 100-lb. rails, teemed in a mold 19 in. square on the base, 17 in. square on the top, and 66 in. long. The ingot, poured 50.5 in. long, was well deoxidized, and therefore had a large cavity. The ingot had a volume of 7.4 cu. ft., inclosing a shrinkage-cavity of about 128 cu. in., practically 1 per cent. of its volume. This is a larger percentage than would be found in rail-steel not so well deoxidized, or which contained numerous blow-holes.

Fig. 2 is a photograph of the bloom of an ingot of the same heat and length, cut for the 9-per cent. mill-discard. The ingot, after stripping and a subsequent ride of 500 ft., was charged directly into the reheating-furnace without allowing the temperature to fall below the recalescence-point, while the bulk of the steel was several hundred degrees above, and in about 2 hr. the ingot was drawn and bloomed. The cavity was small and less than one-tenth of that of the cold ingot of the same heat. I have had a number of ingots and crops cut in recent years, since large planers have been available in the machine-shops. In former years I was obliged to rely upon ingots broken as "stickers" at the drop for a view of the shrinkage-cavities in those for rails. Professor Howe did not mention this advance in practice from the early days of Bessemer, nor have I seen it reported by other authorities.

Fig. 3 is a photograph of an ingot teemed in a similar mold, but poured 53.5 in. long for a 19-per cent. discard, and of a still greater degree of deoxidation than the steel in the ingot shown in Fig. 1. The ingot was slightly inclined when the photograph was taken, and the top does not show apparently as large as in Fig. 1, but it is of practically the same size. The shrinkage-cavity amounted to 250 cu. in., or 1.4 per cent. of the volume of the ingot. The cavity is not bell-shaped or with sides showing parabolic curves, but nearly vertical, by reason of the jolting in the ride to the stripping-machines, then to the scales to be weighed, then to the reheating-furnaces

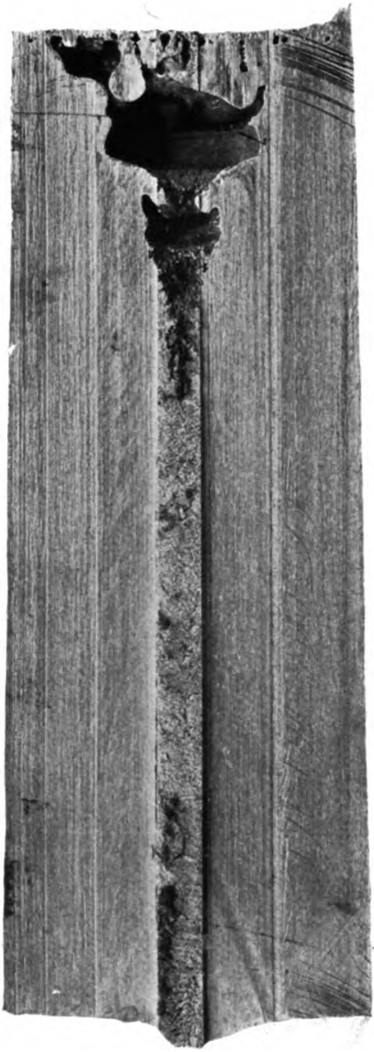


FIG. 1.—SECTION OF INGOT, 17 IN. SQUARE AT TOP, 19 IN. SQUARE AT BASE, AND 50.5 IN. LONG, CONTAINING CAVITY OF 128 CUBIC INCHES.

FIG. 2.—BLOOM FROM AN INGOT OF SAME HEAT AND OF SAME SIZE AS FIG. 1, SHOWING REDUCTION OF CAVITY.

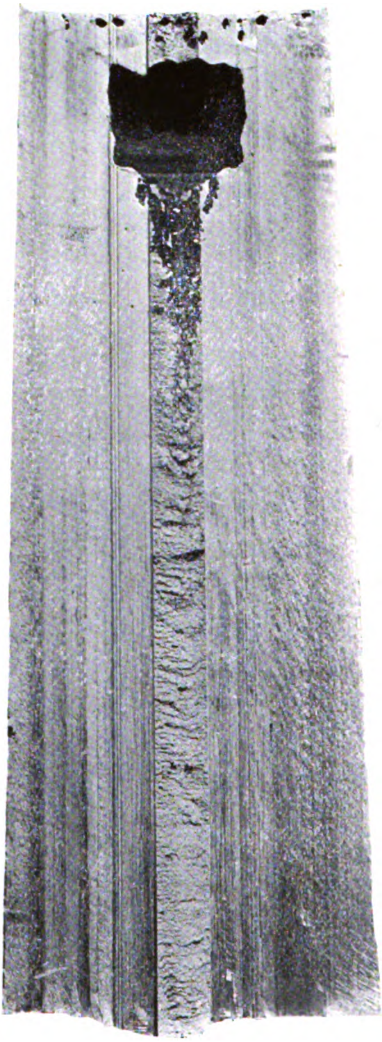


FIG. 3.—INGOT OF SAME SECTION AS FIG. 1, AND 53.5 IN. LONG, CONTAINING CAVITY OF 250 CUBIC INCHES.

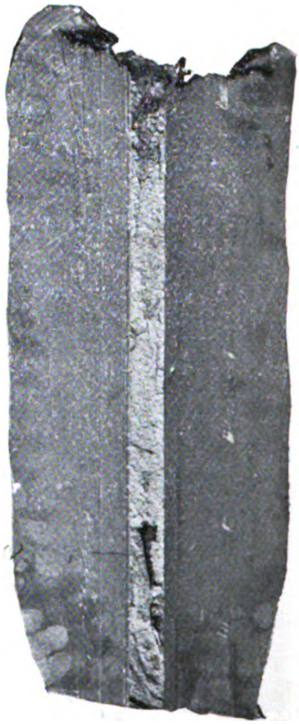


FIG. 4.—BLOOM FROM AN INGOT SIMILAR TO FIG. 3.

holding the other ingots, then 1,000 ft. further to the gantry crane, where it was cooled. It is important to observe that even after its arrival at the gantry the interior was molten under the bottom of the cup-shaped cavity, and a decided shrinkage of the metal occurred below, the walls being lined with pine-tree crystals. The full length of the 19-per cent. bloom-crop could not be cut at one shearing but required two cuts; therefore, a full bloom-crop was not obtained for cutting and subsequent photographing. The cavity was slightly larger than that shown in Fig. 2, though in the bloom it was completely removed in the percentage of discard, and less discard would have answered.

Fig. 4 is from a crop which, by reason of greater deoxidation, is quite similar to one cut from an ingot having the full percentage increased cavity.

The freezing of the stéel for my rails commences on the stool and sides of the molds after a few seconds of contact, and bridges over the top of the ingot in 2 or 3 min. after pouring ceases. Interesting phenomena occur in from 3.5 to 4 min. from the end of pouring, through the expansion of the molds from the sides of the ingot of $\frac{1}{32}$ to $\frac{1}{16}$ of an inch. This expansion of the molds continues until the ingots are stripped in the usual time of manufacture. The exterior of the molds, which measure from 27.5 to 29.5 in. over all, will increase after teeming from $\frac{1}{4}$ to $\frac{3}{8}$ in. per side.

The deoxidizers used seem to make a difference in the quick freezing of the steel, and must be understood and adapted to the practice of the mill in checking the shrinkage-cavities. Aluminum thrown into the molds during teeming, even in small percentages, will often cause the ingot to pipe deeply, while in other cases the pipe will be separated by solid metal into two or more parts. Some of the badly-piped rails which developed in the track during the past two or three years, after about that period of service, were those in which the manufacturers had used aluminum to make "wild" steel set "dead" in the molds without reducing the amount of the other deoxidizers specified by the purchaser. It gives a quicker-setting steel of greater viscosity and apparent shrinkage, which must be stripped and charged into the reheating-furnaces in less time after teeming than the same grade of steel without the alumi-

num, in order to avoid large shrinkage-cavities. In a number of experimental tests made to study the increased shrinkage, it was only with extreme care that it was possible to avoid piping in ingot-molds of the size mentioned, when poured 61 in. long for four 33-ft. lengths of 100-lb. rails. I was then dealing with from 1.2 to 1.5 per cent. interior shrinkage of the volume of the steel, had the ingot been permitted completely to cool. The time of the non-shrinkage or piping period was so reduced that it was not sufficient to avoid piping some of the "A" or top rails of the ingots by the usual conditions of practice which were effectual in holding under sufficient control the shrinkage of ingots for three 33-ft. lengths of 100-lb. rails in the same molds.

The special tests mentioned were made to study the distinction between true piped rails and "split-heads," reported by the trackmen as piped rails, and not as easily detected in the manufacture. My investigations show, I think, that the majority of the "split-heads" with which I have had to deal in the track have occurred in the central core of segregated metal in the heads and webs of the rails. The central core is capped in the bearing-surface by a layer of metal rolled from the exterior of the ingot and often containing slag inclusions. The heads of these rails as manufactured are solid and cut solid by the saws, the inspectors at the mills pass them as sound rails, which they are to all ordinary observation. Laid in the track the metal in the bearing-surface of one or more portions in the length of the rail is inadequate to sustain and distribute the wheel-loads without spreading. Although this takes place only to an infinitesimal amount for each passing wheel, the increment is cumulative, and the rail-head is deformed and eventually splits, after one or more years' service. Etching the top crop would indicate segregation, and often, but not always, shows streaks of cast-iron cut out of the stool by the hot stream of metal in teeming the ingot. This infusion of cast-iron I have found in most of the split-heads which have been investigated in full detail, and is the disturbing factor in many cases of decided segregation. I have found the streaks in a 0.50-per cent. carbon rail to range as high as from 0.80 to 1.02 per cent. of carbon by combustion, though from 10 to 15 points lower by the color-test. The absence or presence in rail-steel

of carbon streaks from cast-iron cut out from the stools or molds has been a disturbing element in studying segregation from the analysis of drillings from different parts of the ingot. It is not uncommon to see in a double stool of an ingot-car that under one mold only a trace of cast-iron has been burned from the stool, while from 30 to 40 lb. has been removed from the other and distributed in the steel of some of the ingots.

The decided segregations when they form part of the head with the cast-iron infusions are often unable to sustain the traffic until the rail is removed for wear, but fail by detailed fractures as a split-head, as already described. When the segregated portion forms the lower part of the web and base, breakages or detailed fractures of the rail as a girder often occur.

I have related briefly some of my observations and practice of many years' experience in the manufacture of heavy and stiff sections of rails. Since the publication of Professor Howe's paper I have tried to confirm his theory of virtual expansion of the walls of the ingot by measurements. Professor Howe in his paper says: "If, for instance, on reaching a temperature of 1000° C. the virtual expansion were such that the ingot was 1 in. wider," etc. I did not expect in rail-ingots to find an increase of any such amount, which for the ingots 19 in. square on the base would augment them to 20 in. square, an increase of 39 cu. in. for each inch in length of the ingot, and for those 50 in. in length of 1,950 cu. in., a volume from 8 to 10 times greater than the cavities found in the ingots cut, which were well deoxidized. Calipering hot molds and ingots as soon as stripped was attended with so many variations of temperature of the molds and ingots that approximate measurements did not definitely indicate an expansion of the ingot-walls. Though all of the molds were made from the same drawings, and were alike for manufacturing purposes, yet each was of a different size, and each was again modified by its temperature when the ingots were teemed. The cold ingots from the same class of molds also varied as to precise size.

In open-hearth ingots showing ejections of sparks and boiling of the steel against the sides of the molds, the metal often rises 3 or 4 in. and "reams in" before it sets on top. This is a virtual rising of the metal on the top of the ingot, producing

spongy steel, and sometimes a cavity is formed when the ingots cannot be charged promptly after stripping. The contraction of solid steel above the critical points is at a faster rate than below. To cut a 33-ft. 100-lb. rail, the saws are set (for rolling at 1,000° C.) at 33 ft. 6.75 in. Calculate the contraction from figures obtained below the critical points, and it is only about one-half of the above amount. The roll-designer in making a hot template allows a contraction of $\frac{3}{16}$ in. per foot.

Molten to frozen steel, when well deoxidized, seems to have a still higher rate of shrinkage as affecting the respective volumes, and it is important to control the temperature-lag as much as possible in teeming and reducing ingots to solid merchantable forms. There is much which can be effectively done in a practical way. During the past year one steel company, for its rail-ingots as soon as teemed through a 1½-in. nozzle, throws on the top of the steel a shovelful of from 3.5 to 4 lb. of coke-dust, which ignites, keeps the molten steel fluid in the center of the top of the ingot, and feeds the shrinkage-cavity. Watching the tops of those ingots for 10 or 12 min., I was never able to see a sudden lowering of the molten steel, as would occur in case of a rapid virtual expansion of the walls of the ingot. Those ingots were stripped in from 30 to 35 min. after teeming.

There has been more attention paid to producing better ingots for rails in the past two years than previously, in the great demand for quantity. The changes in rail-sections by the railroads did not improve the quality or blooming of the ingots. Sink-heads have been tried experimentally and the shrinkage-cavity reduced. Bottom-pouring for open-hearth ingots has been introduced with decided success, and is promised for rail-ingots in a short time.

The principle of making Bessemer steel for quality instead of for quantity was required for all 1908–09 rails on the New York Central Lines—a return to former practice, the beneficial results of which are already apparent.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Conservation of Coal in the United States.

Discussion of the paper of Edward W. Parker, presented at the Spokane meeting. *Bulletin No.* 35, November, 1909, pp. 1011 to 1018.

W. L. SAUNDERS, New York, N. Y.:—Mr. Parker's paper, though entitled Conservation of Coal, might also be called the Conservation of Life in the Coal-Mines of the United States. No subject is of greater importance to mining-men at the present time than information from experts as to how to save coal and how to save human lives in mines. That the casualties in the coal-mines should exceed 3,000 in the year 1907 is simply appalling, and that 1,000 men should have been killed in a single year through explosions alone, and that so good an authority as Mr. Parker should say that "a prolific cause" is an "improperly-prepared blast" or "the failure on the part of the miner to undercut his coal," points to the importance of activity not only among mining engineers but also, through experts, by the legislatures of the respective States.

There is but little doubt that nearly all the serious coal-mine explosions which have taken place in the United States during the past 10 years have been due to coal-dust alone, or coal-dust and gas mixed, and the ignition has been caused by blown-out shots; after very thorough investigation by State Mine Inspectors and Special State Commissioners, this has been fully proven in the large majority of instances. If, then, the blown-out shot is such a deadly agency, and the direct cause of the death of so many thousands of coal-miners, it is natural to ask whether or not such things are preventable, and if they are preventable, how? If the hole is drilled in the proper place to the proper depth, charged with the right amount of powder, and properly tamped or stemmed, a blown-out shot is an impossibility, providing, of course, the coal has been properly prepared for blasting. If then the application of ordinary knowledge, of ordinary skill and experience would eliminate this frightful danger, why is it not done? Because the modern mine-crews are largely made up of men who are inexperienced, unskillful, and densely

ignorant. These men are allowed to drill their own holes; to charge and fire them, notwithstanding that the lives of all the men in the mine are depending on the good judgment of each individual man. At many places all the men have to be out of the mine before the shots are fired, and this dangerous work is performed by shot-firers. In the State of Illinois there is a law which compels this precautionary measure. How dangerous this occupation is may be inferred from the fact that during 1907 12 shot-firers lost their lives by explosions in Illinois. Had the miners been at work when these explosions occurred the loss of life would have been frightful.

That State produces more than 40,000,000 tons of coal per annum, and in 1907, out of a little more than 40,000,000 tons mined, nearly 25,000,000 tons were blasted from the solid; the bill for powder amounted to $2,208,343, and represented 1,261,910 kegs, almost enough to make one suspect that the coal-operators of that State are in league with the powder manufacturers of the United States.

It is interesting to notice that where the coal was undermined by machinery, each keg of powder produced 96.02 tons of coal, while from the solid each keg of powder blasted only 25.78 tons. This comparison needs no comment.

Perhaps the simplest description of a blown-out shot is one that does no useful work in shattering or blowing-down the coal, but blows out its tamping and projects a long vivid tongue of flame into the chamber where it is fired, the floor and sides of which are usually covered with coal-dust; this coal-dust is raised in clouds by the concussion, and in this diffused condition is easily ignited, and an explosion occurs which goes through the mine with inconceivable rapidity, carrying with it death and destruction of property, the extent and violence depending on the amount of dust and good air (oxygen) in the mine. If 1 or 2 per cent. of fire-damp (CH_4) is present in the air, the dust ignites more rapidly. Until recently it was a disputed question as to whether coal-dust could be exploded in the absence of fire-damp, but this question has been settled beyond controversy by the physical tests which have been made, both in this country and in all the coal-mining countries of Europe, in which thousands of people have seen coal-dust without any admixture of gas exploded by using a cannon-shot to represent

the blown-out shot, and an iron tube 100 ft. long and 6 ft. in diameter to represent a mine gallery; coal-dust is strewn on the bottom and on shelves along the side, the cannon loaded with black powder and stemmed with fire-clay is fired into the tube, and a terrific explosion occurs. This has been done hundreds of times and settles the question for all time.

It has also been fully demonstrated by this same method that there are many explosives which do not ignite coal-dust as readily as black powder, on account of the very much shorter flame produced, and these are consequently much safer to use in mines; but as no explosive is flameless, this, while it will prove a mitigation, does not promise perfect immunity by any means.

A blown-out shot occurs when the tamping is the path of the least resistance—this is likely to happen when the hole is drilled beyond the undermining, or when it is drilled on the rib, away from the undermining, or where blasting from the solid is practiced entirely. In some cases it is the result of carelessness; in many the result of ignorance; in some of neither, but of improper preparation of the coal. In thick seams of coal, where the undermining is done with chain-machines, where the undercut is 6 or 7 ft. deep and 4 in. high, the hole is generally drilled in the rib at such an inclination that it will touch or nearly touch the roof at the back. In this case the coal is not properly prepared; it is almost as bad as blasting off the solid, in some cases worse, for the way the hole is drilled makes it more dangerous. Many of the most disastrous explosions have occurred exactly under these conditions. It is a significant fact that the most destructive colliery-explosions which have occurred in recent years have been caused by blown-out shots, where the coal has been mined with chain-machines. Harwick, Monongah, Darr, and Marianna are notable examples: in these four explosions nearly 1,000 lives were lost.

The truth is, that undermining coal is not a sufficient preparation for blasting; it should be sheared on one side or in the middle; in this condition less than one-half the powder would be necessary, less than one-half of the smoke would result, giving better sanitary conditions, and less danger of falling roof, which is largely caused by excessive blasting, and is a prolific cause of injury and death. All the coal would be in a better

condition for handling and rehandling, giving very much less slack at destination, and a positive gain of at least 20 per cent. more lump, and, in addition to all this, immunity from blown-out shots, from destruction of life and property.

The value of shearing in the preparation of coal is recognized by mining engineers, mine-superintendents, and mine-foremen, as well as coal-operators, but because they think it adds to the cost of production, they are willing to forego all the many advantages which it gives. It would add little, if any, to the cost of coal, for when lump coal is worth from $1 to $1.40 per ton and slack coal is worth from 20 to 40 cents per ton at the mines, and shearing will increase the more valuable coal by 20 per cent., and the cost of shearing with approved shearing-machines, would be about 6 cents per ton, it is not hard to figure what it would cost.

Shearing will be universal some day, and it will be a blessing to the workman and to the operator, and to all interested in the conservation of our national fuel-supply.

It is likely that in a comparatively few years the vast bulk of our bituminous coal will be produced without explosives of any kind; it will be excavated and loaded by mechanical means entirely, reducing explosives to a disappearing minimum, and so reducing accidents by falls of roof that a death through this cause will be as rare as it is now frequent.

A great many more deaths result from falling roof and coal than from all other causes combined, but little notice is taken of them because the fatalities occur only one or two at a time, and are scattered all over the coal-regions, wherever there are coal-mines in operation. This great loss and waste of life should not be less appalling because there is nothing spectacular about it—the sorrow and suffering are just as acute, there are just as many widows and orphans, just as many bereaved fathers and mothers, as if the lives had been lost in explosions, and just as earnest efforts should be made to prevent deaths from this cause as from any other. It is easy to say that in most cases it is carelessness or ignorance, or both, but this does not relieve us from responsibility, especially if ignorance is the cause, for if we employ workmen who are ignorant it is our duty to teach them, and so to safeguard their lives and others' that they shall not lose them through ignorance; and much can be done in this

way, and hundreds of lives saved every year, materially reducing the disgraceful list of deaths through this cause. Fully 95 per cent. of all fatal and non-fatal accidents caused by falling slate or coal take place in the working-faces where the miners are engaged in digging or loading the coal. In some mines the roof is naturally bad, and should be carefully and systematically timbered, the props having over them stout, broad cap-pieces, presenting a wide surface to the roof and set as near the face as necessity may demand. If this is done the miner can work between the props with perfect safety; but as he looks on this work as being unproductive, he will not take the time, or he may feel indolent and neglect to make his working-place safe. In order to overcome this, there must be stringent rules compelling systematic timbering after an approved method, which would make all working-places safe; it might be objected that some places do not need as much timbering as others, but the answer to this is that where there is such ignorance, all places must be looked upon as dangerous. One of the most prolific causes of falls of rock and slate in the faces is blasting, especially where the coal is undermined by chain-machines, or blasting off the solid is practiced, as exceptionally large shots are necessary; these shots jar the roof to such an extent that the roof, which before was comparatively safe, becomes loose and dangerous, and too often the miner goes into a place which has been thus rendered unsafe and commences to load coal under a roof which may come down, and frequently does come down, maiming him or crushing out his life. At least 75 per cent. of these accidents can be avoided if the mine-managements adopt proper methods which are rigidly enforced. As already indicated, one method is in systematic timbering; if in addition to this all blasting from the solid is made a criminal offense, and blasting is permitted only when the coal has been undermined and sheared on one side, then the roof would sustain but little, if any, injury from powder-shocks. If this precaution were observed, and a man were employed to visit each place once (or, if necessary, twice) a day, making sure that all the rules are enforced, deaths by falling roof would be rare, instead of daily, occurrences. In the North of England one man, who is called a Deputy, is employed for every 20 or 30 men. His business is to look after their safety,

to visit every place as often as necessary, each shift, to see that each place is amply supplied with timber of proper length, and in cases of peculiar danger to set it himself; he carries an axe and saw, and is always ready for emergencies. These miners are to the manner born; they come of long generations of miners, and in all the world there are none more skillful or intelligent, all speaking a common language. If it is necessary to throw such safeguards around them, how much more necessary it is where we have such a lack of skill, such dense ignorance, and so much difficulty in oral communication.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Dust-Explosions in Coal-Mines.

Discussion of the paper of Mr. Bache, presented at the Spokane meeting, September, 1909. *Bulletin No.* 32, August, 1909, pp. 741 to 748.

R. W. Raymond, New York, N. Y.:—I think Mr. Bache has put his finger on the chief source of the danger of dust-, or gas-and-dust, explosions in collieries. I mean the persistent determination of the miners' unions to increase their weekly wages by the excessive use of explosives. This would not be feasible if coal-miners were paid by the day; but this form of payment is, for many reasons, not economically practicable; and the universal practice is to pay for the winning of coal according to the quantity produced. If the miner, by using a large amount of powder, can throw down a large amount of coal without corresponding labor on his own part in under-cutting and drilling, he will receive more money for less work, provided he is paid for everything—merchantable coal, worth less dust, slate, and "bone"—resulting from such a method.

My attention was called to this matter many years ago by an admirable report of Prof. W. B. Potter, a past-President of the Institute, on the conditions obtaining in this respect in the Illinois coal-field. It was made very clear in that report that considerations of danger to workmen or loyalty to employers could not be relied upon to prevent miners from employing this perilous and wasteful method of increasing their own immediate receipts.

So far as I know, only three remedies have been attempted, namely: the enforcement of discipline as to the methods of mining; the refusal to pay for dust, etc., produced by the miners' methods; and restriction upon the use of explosives, effected by requiring the miner to purchase them from the employer, at a price so high as to make it unprofitable for him to use them in excess. All of these attempted remedies have encountered the bitter opposition of the miners' unions. The enforcement of discipline has become almost impossible, if discipline be (as it should be) understood to involve punishment

for the violation of rules when no disaster has followed. I know of a comparatively recent case in which a miner, in a highly "fiery" colliery, was found to have matches in his pocket, contrary to express rule. But the superintendent did not dare to discharge him, knowing that a strike would immediately follow, and that, even if the committee of the Union should ultimately decide that the discharge was justifiable, and should order the men back to work, there would be a delay of one or two weeks in the operation of the mine, the cost and loss of which would fall upon the company. A week or two later there was a fearful explosion in the same colliery, destroying many lives. And I saw afterwards, in a respectable journal, the statement that this loss of life was chargeable to "the greed of capital."

The second remedy—namely, the refusal to pay the miner for worthless material, has been a fruitful source of trouble. It was hardly to be expected that the miners' unions, maintaining an attitude of war towards employers, and regarding every interval of peace as simply an armed truce, would regard as just the deductions made by officials from their car-loads of miscellaneous stuff. Though forced to recede from their original demand to be paid for everything, they still protested against unfair treatment; and this system, it must be confessed, left room for such complaints.

The third remedy, adopted after much consideration and experiment in the anthracite-regions, proved the best of all. The miner was required to buy explosives at a price (higher than the market-price) which made it unprofitable for him to substitute powder for labor. At the same time, the extra cost of the powder which he did use was taken into account in fixing his remuneration, so that he remained, after all deductions, the best paid of all laborers of his class.

Unfortunately, the true meaning of this arrangement seems not to have been brought clearly to the attention of the "Roosevelt anthracite commission." It was widely misrepresented as an attempt on the part of the operators to squeeze out of their employees a miserable extra profit; and the operators, anxious above all to avoid the continuance of disastrous conflict and the weight of popular odium, surrendered the point without adequate defense. So the Commission anni-

hilated by its report the result of many years' study of the subject, and the most acceptable and effective automatic remedy for a great evil—a remedy which had operated successfully for years before it was thus summarily discredited. I fear it will be long before we succeed (if we ever do succeed) in restoring an arrangement so satisfactory.

Mr. Bache points out another illustration of the way in which public sympathy is utilized for special interests. I refer to the statement in his paper (p. 743), that mining companies and State legislatures are influenced by humane considerations to provide that blasts shall be fired by persons specially appointed—in other words, that the miner who has prepared a dangerous shot shall be forbidden to fire it himself! There is reason to believe that this provision has been utilized in some instances for purposes of private revenge—the hole being bored to extra depth and overloaded and the fuse so arranged as to "get" the shot-firer, against whom the miner had a grudge. A simple remedy for this evil would be to ordain that the man who fires the shot should load the hole.

The official Federal investigations as to the possibility and the peril of explosions and mine-fires due to the presence of coal-dust have confirmed similar investigations abroad, and given us much interesting information about our own conditions. But they will not develop the chief danger until they expose fearlessly the causes which Mr. Bache has indicated.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Pan-Amalgamation: an Instructive Laboratory-Experiment.

Discussion of the paper of Messrs. Hofman and Hayward, presented at the New Haven meeting, February, 1909, and published in *Bulletin No.* 30, June, 1909, pp. 513 to 529.

E. A. H. TAYS, San Blas, Sinaloa, Mex. (communication to the Secretary*):—The results obtained by Messrs. Hofman and Hayward in their experiments, proving that a low percentage of copper sulphate with a variable percentage of salt, depending on the ore, gives the best results, confirm practical mill-work which I did in 1895. I have none of my notes, taken at the time, to refer to, so have to rely solely on memory, which precludes the conciseness that is alway desirable.

In 1894 I took charge of the plant of the San Rafael Company, near Chinipas, Chihuahua, Mex., which at that time was treating a very refractory ore carrying about $15 in gold and from 20 to 25 oz. of silver per ton, the bullion being about 300 fine in gold. The silver in the ore was in the form of a sulphide.

Although carrying an average of 0.75 oz. of gold per ton, free gold could rarely be seen by panning, but upon roasting a sample of the ore the gold became visible at once—a result which was discovered by chance and used to advantage later.

In order to utilize the old mill to the best advantage, it was decided to use pan-amalgamation after roasting. The roasting, done in a lime-kiln, was similar to the ordinary "burning" operation. The roasted ore was trammed to the mill-bins and crushed by 10 stamps through a 30-mesh screen. The pulp was de-watered and fed to four 1-ton pans.

My recollection is that upon starting the mill-work, the charge contained 1.5 lb. of copper sulphate and 5 per cent. of salt per ton.

* Received July 12, 1909.

From the start the extraction was unsatisfactory and the loss of quicksilver was very large. Experiments were then carried out on a working scale. First, the quantity of copper sulphate was increased 4 oz. at a time; then it was reduced at the same rate and careful notes recorded. To my surprise, when the charge of copper sulphate was lowered from the original 1.5 lb. per ton, the percentage of extraction increased, and at 4 oz. per ton of charge the best extraction was made. The quantity of salt was then varied, and 5 per cent. was found to give the best results with the 4-oz. charge of copper sulphate.

When I explained what I was doing to the metallurgist of a nearby plant, he asked, "Why do you use any copper sulphate at all?"

The nature of the ore may be judged by the fact that with only 4 oz. of copper sulphate per ton of ore treated the loss of "quick" was still 1 lb. per ton of ore treated.

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BLUE ENAMELED PIN

GOLD CROSS HAMMERS

SEAL FOB